PREFACE

Flotation concentration includes within its scope almost as many processes as all other methods of ore concentration combined, the only elements common to all the processes being selection, or concentration, and separation of the concentrate from the tailing by flotation of the former.

Widespread understanding of the physical principles underlying flotation phenomena and of the diversity of flotation processes has been delayed for divers reasons. The apparent complexity of the phenomena and the difficulties of investigation are sufficient to explain some of the delay, but much of it is chargeable to the stand of patent-owning corporations in their attempt to establish a monopoly on flotation processes. These companies have steadfastly opposed dissemination of knowledge of the art by their employees and licensees, notwithstanding the moral and legal duty of a patentee to make full and truthful disclosure of all he knows concerning the subject matter of his patent; by threats of litigation sown broadcast they have succeeded in causing a veil of secrecy to surround the operations of non-licensees; and by their unfounded claims that all flotation processes prior to that described in U. S. Patent 835,120 were laboratory curiosities or commercial failures, and that those subsequently discovered were merely improvements of that process, they have caused the spread of wrong ideas on the part of many of those interested.

It is the purpose of this book, in part, to counteract the further spread of false conceptions concerning flotation concentration, by setting forth some of the essential facts which contradict them; in part to describe apparatus and methods of testing which will aid investigators in their own researches; finally, to give some generalizations from mill practice, by means of which the labora-

tory experimenter can translate his results into commercial operations.

The experimental data upon which the distinction between pulp-body and bubble-column processes is based were collected under the author's direction in early 1919 and those upon which the essential condition of supersaturation of the liquid of the pulp with gas in pulp-body flotation concentration is predicated, were obtained in further investigation of differences in the processes, in the latter part of 1919 and early in the present year. Subsequent to the latter discovery the author came into possession of a copy of "The Theory of 'Concentration Processes Involving Surface Tension,'" by H. Livingstone Sulman and Hugh Kirkpatrick Picard, mentioned in Hoover's book.* These gentlemen are two of the patentees of U. S. Patent 835,120. Their treatise shows that they, as early as 1907, recognized not only the essential similarity between their own and the prior pulp-body processes, but also the importance, in these processes, of the condition of supersaturation of the liquid of the pulp with gas. This treatise has never been published. A paper on flotation was published by Sulman in 1919 † in which no mention of the similarity stated in the treatise is made and in which supersaturation is never touched upon.

Description of the experiments upon which the conclusions stated in Chapter I are based has purposely been omitted, as suitable presentation of these experiments demands discussion which is unsuitable for a laboratory manual. It is planned to present these and other data in a later book which will attempt to treat of flotation exhaustively. The text of the other chapters has been condensed as far as was, in the author's opinion, compatible with clearness, in the hope that such brevity would make the information contained more easily and quickly available.

Thanks are due many mill men and manufacturers for their generous and cheerful aid in furnishing information. For data taken from the literature acknowledgment has been made in the text and footnotes.

* "Concentrating Ores by Flotation," The Mining Magazine, London, 1916.

† "A Contribution to the Study of Flotation," Bull. I. M. M., Nov. 1919.

The pulp routing in many of the mills is now, to the writer's knowledge, different from that shown in the flow sheets presented, but, since these flow sheets were chosen only to illustrate types and not to serve as rigid guides, the variation is unimportant.

ARTHUR F. TAGGART.

NEW HAVEN, CONN. *October* 1, 1920.

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A MANUAL OF FLOTATION PROCESSES

CHAPTER I

INTRODUCTION

Flotation, as the term is applied to ore concentration, means the separation of one of the constituents of an ore from the remainder by causing it to float at or above the surface of a pulp consisting of the finely pulverized ore and water.

Minerals that float have a metallic, adamantine, or resinous luster. Minerals with vitreous, pearly, or earthy luster do not float, as the term is at present used in the art of concentration. It must not be understood, however, that the float concentrate in a flotation operation is free from these latter minerals. As a matter of fact, in many flotation concentrates the minerals of non-metallic luster predominate. Their inclusion is due in part to their being mechanically entrapped and held, as on a screen, by the bubbles composing the froth, in part to the inclusion of pulp in thick bubble walls, and in part to the removal of some of the pulp, as such, with the floating concentrate.

Ores amenable to concentration by flotation. Almost any ore consisting of a mineral of metallic, resinous, or adamantine luster associated with the usual rock-forming minerals, can be concentrated by flotation. In the great majority of cases the part of the ore that floats is the valuable portion, but if the constitution of an ore were such that the valuable mineral had a non-metallic luster, and the gangue a metallic * luster, the floated portion would constitute the tailing while the valuable portion would remain submerged and be drawn off as residual pulp.

* Throughout the following discussion the phrases "mineral of metallic luster," "metallic mineral," "metalliferous mineral" and "sulphide," or "sulphide mineral" will be used interchangeably, as is the usual case in flotation terminology, to designate minerals of metallic, adamantine or resinous luster.

The grade of concentrate, the ratio of concentration, and the percentage of recovery will depend upon the ore itself and the method of treatment employed. Thus a copper ore consisting of chalcocite in a gangue of rock-forming silicates will yield, under a given method of treatment, a higher grade of concentrate and show a higher ratio of concentration than another copper ore of identical copper content, in which the copper mineral is chalcocite, but in which pyrite is associated with the other gangue minerals. This, for the reason that the flotation methods ordinarily practiced do not differentiate to any considerable extent between the minerals of metallic luster in the ore, but bring up all such minerals in the concentrate.

Methods of flotation may be classified for purposes of study into three varieties, on the basis of the force that acts to buoy the mineral of metallic luster, as: skin-flotation methods, oil-flotation methods, froth-flotation methods.

Skin-flotation methods depend for their operation on the comparative reluctance of the minerals of metallic lustre in an ore to become wetted by water and the resultant buoyant effect of the force of surface tension exerted on these particles at the upper surface of a body of liquid pulp. Apparatus for the practice of skin flotation is of one of two general classes, depending upon whether the ore is fed to the machine dry or wet. Crude methods employing dry feeding have been rather widely used in the graphite industry. The most elaborate apparatus employing dry feeding is that described in U. S. patent 1,088,050 issued to H. E. Wood, February 24, 1914, and described on page 104. The best known of the methods in which wet ore is presented to the flotation machine are the Macquisten and DeBavay, which are described on pages 106 and 108 respectively. In general, the methods that feed dry ore require the dust to be separated before the ore is fed to the machine on account of the fact that very finely divided gangue is as difficult to wet at the surface of a body of water as is the mineral of metallic luster. This limitation as to size is not so important in "wet" skin-flotation methods. Skin-flotation produces a high-grade concentrate at the expense of a low recovery. On lead-zinc ores, differential flotation of a part of the lead in the form of a high-grade lead

concentrate may be accomplished. This fact is, in some cases, held to justify the use of such methods preceding froth flotation. Otherwise the use of skin-flotation methods at present is limited to the case where grade of concentrate produced is a more important consideration than the recovery obtained.

Oil-flotation methods effect the selection of the minerals of metallic luster from the gangue minerals in an ore by reason of the fact that the minerals of metallic luster are wetted preferentially by oil in the presence of water and, hence, pass from the aqueous pulp containing them into the oil, or, more accurately, into the interface between the oil and water; while the gangue, with the reverse tendency so far as wetting is concerned, remains in the water. The buoyant effect necessary to float the selected mineral of metallic luster is brought about by reason of the difference in weight between the system of oil effectively acting plus mineral of metallic luster carried thereby and the weight of pulp that it displaces. The metallic mineral is held in the oil by the viscosity and resistance to breaking of the film interfacial to the oil and the water of the pulp.

The processes utilizing oil to select and float metallic minerals are of two varieties, viz.: those in which the oil is mixed with the ore in the presence of little or no water and those in which the ore, before admixture of the oil, has been brought to the condition of a freely flowing pulp. The better known processes of the first class are those of Everson and Robson; of the second, Elmore and Wolf and Scammell. These processes are described on pages 110 *et seq.*

Froth flotation comprises two entirely different types of processes which resemble each other only in the fact that in both the concentrate is removed in the form of a froth composed of gas, liquid, and solid matter preponderantly sulphide mineral. The processes differ fundamentally both in the place in which concentration is done and in the mechanism of the selection of sulphide from gangue. On the basis of the first difference the processes may be classified as pulp-body concentration processes and bubble-column concentration processes.

Pulp-body-concentration processes may be subdivided, on the basis of the method of introducing the bubble-making gas, into

four types: (1) chemical-generation, (2) pressure-reduction, (3) boiling, and (4) agitation. All four types depend upon the fact that in a pulp, the liquid part of which is saturated with a gas, preferential precipitation of the gas on the sulphide particles can be brought about by so changing the conditions of temperature and pressure that the liquid is, under the changed conditions, supersaturated. This preferential precipitation of gas from the supersaturated liquid is enhanced, if the sulphide particles are coated with an oily substance, and the presence of such a substance also makes greater the force of adherence between the precipitated bubbles and sulphide particles. As a result of this preferential precipitation of gas on sulphide particles in the pulp, and its adhesion thereto, there are formed in the body of the pulp agglomerates consisting of one or more gas bubbles with sulphide particles firmly cemented to them. These agglomerates later rise to the surface in the form of a froth which is separated as concentrate. Observation of any of the pulp-body-concentration processes shows clearly this phenomenon of rising agglomerates whose color indicates distinctly that concentration has been completed at the surfaces of the bubbles composing them, *below the surface of the pulp*, that is, within the pulp body.

Chemical-generation processes are typified by the Froment process and the Elmore electrolytic process. The former is described in detail on page 114. In the Froment process, gas is produced in pulp in the presence of oiled sulphide particles by causing sulphuric acid to react with carbonates, either naturally or artificially present. In the Potter-Delprat process as most extensively practised the same method of gas production is employed, but no oil is present. In the Elmore electrolytic process, hydrogen and oxygen are produced by the decomposition of water caused by the passage of an electric current through a pulp in which an electrolyte is present. In all of these processes it seems to be essential that at least a part of the gas pass through the solution stage in order to effect adherence to the sulphide. Gas which is freed in the form of bubbles at the surface of carbonate particles in the pulp and which persists as a bubble in

its passage through the pulp, will rarely, if ever, adhere, in such passage, to sulphide particles. Such bubbles may coalesce with other bubbles already present on sulphide particles and thus aid in flotation, but they play their principal part in providing additional bouyancy in the froth and in picking up the sulphide particles in the froth which are dropped by the breaking of other bubbles therein.

The change in condition effective in these processes to produce local supersaturation is one of "solution pressure." At the surface of the dissolving carbonates there is pressure by the molecules of carbon dioxide evolved tending to drive them into solution in the water. Those which dissolve travel by diffusion and by reason of water currents away from the regions where the forces tending to drive them into solution exist. In these regions of lessened solution pressure the molecules tend to come out of solution, to precipitate, and they do so preferentially at the surfaces where the least resisting forces exist, which, in this case, are the sulphide surfaces.

Pressure-reduction processes depend upon a reduction in external pressure to bring about the condition of gaseous supersaturation essential to preferential precipitation of gas bubbles on the surfaces of sulphide particles. These processes are of two kinds. In one variety the water of the pulp is saturated with air by being subjected to pressures greater than atmospheric. Upon subsequent discharge into the atmosphere the air dissolved under super-normal pressure is released at sulphide surfaces, and the bubbles adhering thereto eventually raise the sulphide mineral to form a froth. The Norris patent, U. S. 873,586, is the most promising of this group, which includes U. S. patent 835,479. The other variety of pressure-reduction process is typified by the vacuum process invented by F. E. Elmore and described in U. S. patent 826,411. A detailed description is given on page 114. In the vacuum process a pulp pre-mixed with oil is subjected to a vacuum, which causes the air contained in the water to come out of solution. The air coming out of solution does so preferentially at the surface of the minerals of metallic luster and adheres thereto. When the sys-

tem of bubble and sulphide has become sufficiently buoyant to rise, the sulphides are carried to the surface to form a froth.

Boiling processes depend upon heat to drive air and water vapor out of the water in a pulp. These gases form bubbles preferentially at the surfaces of the metalliferous minerals and the bubbles with their solid load rise to form a froth. U. S. patent 835,143 is typical of this type of process. The phenomenon is effective both with and without "oil" and actual boiling is not essential.

The agitation-froth process depends upon local supersaturation of the water of a pulp with air by the mechanical action of a swiftly revolving beater, and the simultaneous precipitation of air in the form of bubbles, preferentially on the surface of the particles of metalliferous mineral, to effect the same result effected in the previously mentioned processes of the pulp-body-concentration type. Agitation-froth machines are described in detail on pages 115 *et seq.* They consist essentially of an agitation chamber in which a stirrer mounted on a vertical spindle rotates at high speed, and a froth-separating compartment in which the pulp is allowed to come to rest and the bubbles carrying the metalliferous mineral rise to the surface to form a froth which is skimmed off. The pulp in the agitating compartment, under the influence of the rotating agitator, is thrown from the center toward the side of the chamber. The result is that the surface of the pulp assumes the shape of an inverted cone. When the cone becomes sufficiently pronounced that the tip reaches down to the revolving beater arms, the tip is cut off and a large bubble of air is entrained. This bubble is immediately broken up by the direct impact of the impeller arms and by the violent swirling of the pulp into a large number of small bubbles. These bubbles, due to their minute size, are in the most favorable state to be taken into solution, and many of them are, at the time of their formation, subjected to considerably more than atmospheric pressure, due to the impact of the impeller blades. They have, also, on account of their small size, but slight vertical motion relative to the pulp mass, and are, therefore, kept for a comparatively long time in contact with the water and subject

to the impact of the impeller blades. As a result, some of them go into solution in the water. At the same time there exists behind each impeller blade a volume of pulp on which the pressure is reduced by reason of the forward movement of the blade and the inertia of the pulp mass. Here air comes out of solution in the form of bubbles at the surfaces of the sulphide particles. The excess bubbles which never go through the solution stage, in this, as in the other pulp-body-concentration processes, in part coalesce with the bubbles already formed on sulphide surfaces; in part pass with the pulp into the froth-separating chamber and there, rising, add buoyancy to the froth and serve to pick up particles dropped by the bursting of other bubbles; in large part, however, they rise to the surface of the pulp in the agitating compartment and are lost to the process.

The froths produced in pulp-body-concentration processes are small-bubble, coherent and persistent, and characteristic. The volume of gas effectively utilized in floating the mineral is of the order of 20 to 50 cu. ft. per cu. ft. of solid floated.

In the bubble-column process substantially all of the concentration is done in a column of bubbles above and floating on the surface of the body of pulp. In this process the volume of gas effectively used to produce concentration is enormously greater than in pulp-body concentration, being of the order of 1000 to 2000 cu. ft. per cu. ft. of solid floated. The result is that the froth is fragile and evanescent and strikingly different from that characteristic of the other class of processes. Further investigation of the process, by observation of the operation in glass-sided machines, makes apparent the following facts: (1) The bubbles are much larger than in pulp-body processes; (2) they are more numerous; (3) they rise through the pulp more rapidly; (4) they arrive at the surface of the pulp with a solid load composed of sulphide and gangue in the same proportions that these exist in the pulp through which they have passed; (5) concentration begins at the bottom of the bubble column (i.e., the surface of the pulp body) and progresses upward. The actual mechanism of the concentration itself can be observed by studying the bubble column with a hand glass. Such study

shows that in the bubble walls there is a differential draining of the gangue and sulphide particles; that the average downward velocity of the sulphide particles is less than the average upward velocity of the bubbles; that the average downward velocity of the gangue is greater than the average upward velocity of the bubbles; and that, as a result, the sulphides are lifted up and away from the gangue. It is apparent, also, from such study, that the sulphide particles in the bubble column are nowhere firmly adherent to bubbles, as they are in the pulp-body processes.

Machines in which the bubble-column process is practiced may be classified, on the basis of the method of introducing air, as plunging stream or cascade machines, pneumatic machines and centrifugal machines.

In plunging-stream type bubble-column machines, air is carried into the pulp by a plunging stream of pulp. The bubbles are relatively large and the disturbance of the pulp body relatively slight. Hence there is quick rise of relatively large bubbles through the pulp which does not cause supersaturation with subsequent precipitation and concentration below the pulp surface, but rather necessitates that such concentration as takes place shall occur in the bubble column. One type of plunging-stream machine is described on page 133.

Pneumatic bubble-column machines are typified by the Callow cell, which is described on page 122. In this device air is introduced into the pulp through a porous medium. Canvas, cotton twill, blanket, carborundum, concrete and other porous substances are used as media for the distribution of the entering air. In pneumatic machines the pulp is relatively quiescent, the bubbles are larger than in agitation-type machines and hence rise rapidly. No pressure is exerted to force them into solution, nor is there any local release of pressure to cause air already in solution to precipitate. The result is that no selection of sulphide particles takes place beneath the pulp surface. The bubbles rushing upward through the pulp mechanically push a certain amount of pulp above them as they emerge, with the result that the walls of the emerged bubble contain a solid load of the same composition as that in the body of the pulp. At

the pulp surface the speed of rise of the bubble abruptly lessens and the solid particles which now form a part of the bubble film begin to drain away rapidly. At the same time the bubble is lifted by the bubbles which follow it to the pulp surface. The solid particles drain away at different rates, the gangue particles much the more rapidly, so that, if the air supply and consequent rate of rise of bubbles is properly adjusted, the average downward velocity of the gangue will be greater than the average upward velocity of the bublbe, and it will largely settle back into the pulp, while the average downward velocity of the sulphide will be less than that of the bubble, with the result that the sulphide will be carried up and away from the gangue and may be separated as concentrate.

Centrifugal bubble-column machines include several in which air is drawn into the pulp due to the revolution of a disk or impeller on a vertical shaft and two in which the air-entraining mechanism is mounted on a horizontal shaft. In the former class are the Ruth and Groch hollow-shaft machines and the Hebbard sub-aëration machine. The Rork and K. and K. (Kraut and Kollberg) machines are of the horizontal-shaft type. These machines are described on pages 135 *et seq.* They appear on first inspection to be of the agitation type but study of their action shows that pulp is displaced so rapidly through the zone of the moving parts into the quiet zone that the time is insufficient to effect pulp-body-concentration. This conclusion is confirmed by study of the bubble column in the machines. Such study demonstrates clearly that practically no concentration has taken place at the bubble surface by the time the bubble reaches the pulp surface and that substantially all of the concentration takes place subsequent thereto.

Differential flotation is the term applied to methods of concentration by flotation which seek to float one only of the minerals of metallic luster in a mixture of such minerals either in or out of the presence of the usual rock-forming gangue minerals.

Flotation of oxidized ores is a misnomer unless it is understood that the oxidized mineral containing the sought-for metals,

usually lead or copper, is first changed by chemical means into some compound of the metal having a metallic luster. The latter product can then be floated with varying success. In some of these methods the transformation of the oxidized mineral requires complete solution of the metal and subsequent precipitation as metal or sulphide; in other methods a surface transformation only, usually to sulphide, is attempted. No generally applicable and commercially successful method for the treatment of oxidized ores by flotation has yet been devised.

Flotation agents include oils and certain other organic compounds, and many inorganic compounds. All of these substances will act alone to effect concentration by froth-flotation. Usual practice, however, is to use an oil or mixture of oils as the principal agent, with or without the addition of some inorganic substance. The oils commonly used are (1) essential oils, of which class pine oils are most frequently employed; (2) coal-tar oils; (3) wood-tar oils; and (4), petroleums. Sulphuric acid is the commonest inorganic agent; lime, salt cake and soda are less frequently added.

Conditions of operation of froth-flotation. Froth-flotation is in general effective only on ores ground to pass 0.3-mm. aperture or less and the agitation-froth and bubble-column processes are most effectively practiced when the bulk of the feed will pass a 0.074-mm. screen. In these latter processes the finely pulverized ore must be in the condition of a freely flowing pulp with water. The most favorable percentage of solids lies within the range of 15 per cent. to 30 per cent. The operation of the process is affected by the mineralogical character of the ore, the grade of feed, the kind and quantity of flotation agent and, to a less extent, by the temperature of the pulp, the place and method of adding the flotation agent, the type of flotation machine used, and the method of removing concentrate.

Typical flow-sheets in flotation practice. Flotation enters into mill flow-sheets in two capacities, viz.: (1) as a method of treatment accessory to gravity concentration; (2) as the principal method of concentration. Which of these flow-sheets is followed in a given mill should be determined primarily by the

character of the ore, bearing in mind that gravity concentration is more economical than flotation when the valuable mineral occurs in such sizes that it can be suitably treated on shaking-tables or jigs. Hence, when the valuable mineral occurs in the ore in aggregates more than one or two millimeters in diameter and the projected mill tonnage is such that both gravity-concentration and flotation machines can be fed to capacity, a flow-sheet in which flotation is accessory to gravity-concentration should be investigated. Otherwise flotation should be the principal process, with gravity-concentration subordinate, if, indeed it be employed at all.

Step treatment. It is inherent in most froth-flotation processes that a high grade of concentrate and a high percentage of recovery cannot be made in one and the same cell at one and the same time. It is necessary to first treat the pulp in one flotation machine, usually called a "rougher." In this machine the attempt is to produce a clean tailing and a dirty concentrate. The dirty concentrate is then re-treated in a second machine, usually called a "cleaner," which makes a clean concentrate and a high-grade tailing. Common practice is to re-treat the tailing from the cleaner cell. In another scheme of treatment a clean concentrate and a high-grade tailing, or middling, are made on the first treatment cell or cells. The high-grade tailing is further treated in other cells which make a low-grade concentrate and a finished, low-grade tailing. The low-grade concentrate from the later machines is re-treated in cleaner machines as above described, or is returned to the head of the first machine.

Flotation results. Since, in flotation, only minerals of metallic, resinous, or adamantine luster are selected, investigation and judgment as to the efficiency of the process should be based on a consideration of the recoverable mineral only. Thus in an ore containing both sulphide and oxide copper the recovery credited to the process should be based on the sulphide copper content of the feed, concentrate and tailing, if a true measure of the efficiency of the process is sought.

Froth-flotation, properly practiced, will recover from 60 to well over 95 per cent. of the sulphide mineral content of an ore

in the form of a concentrate containing from 10 to 40 per cent insoluble matter, i.e., gangue.

Variables of operation. The results to be obtained in flotation depend, more than in any other concentration process, on the proper coördination of a considerable number of operating variables. These variables are listed below.

Ore —
- Mineralogical character.
- Recoverable mineral content.
- Size of particles.

Principal reagent —
- Character.
- Quantity.

Minor reagent —
- Presence.
- Character.
- Quantity.

Percentage of solids in pulp.
Method of flotation employed.
Temperature of pulp.
Time of treatment.
Recovery desired.
Grade of concentrate desired.

Ore. The mineralogical character of the ore determines, in a way, the character of the principal flotation agent or "oil" to be used. From this statement it is not to be understood that for a given ore one and only one oil can be used or even, disregarding for the moment the commercial consideration of availability, can be best used. But the physical phenomenon upon which selection depends is specific to a considerable extent. By no means all minerals of metallic or adamantine luster are selected from all gangue minerals by a given oily substance, and by no means all oily substances will act successfully in all flotation processes.

The mineralogical character of the ore in connection with the character of the principal flotation agent determines whether

or not another agent is necessary. Thus ores containing a considerable portion of argillaceous matter will not respond successfully to flotation unless this material is, to a considerable extent, flocculated in the pulp. If the oil fails to produce this necessary flocculation another agent, usually an electrolyte, is necessary.

The mineralogical character of the ore determines also the extent to which grinding must be carried before flotation is applied. This matter has been previously discussed.

In the agitation-froth process, the recoverable mineral content of an ore, the amount of a given oil necessary, the percentage of solids in the pulp treated, the grade of concentrate and the recovery attained are strictly dependent variables. This interdependence may be stated as follows:

1. In order to recover a given percentage of the recoverable mineral in an ore in the form of a concentrate of a given grade, if the percentage of solids is fixed, the amount of a given oil necessary is in direct proportion to the amount of recoverable mineral in the feed.
2. In order to recover a given percentage of the recoverable mineral in an ore in the form of a concentrate of a given grade, if the grade of the feed is kept constant, the amount of a given oil necessary is in almost direct proportion to the percentage of moisture in the pulp.

These relations have been proven conclusively for the agitation-froth process and should, therefore, hold for the other pulp-body-concentration processes. Some similar relation is indicated for bubble-column processes, but the writer is aware of no exhaustive and conclusive work in this direction, and the dissimilarity in the mechanism of the two types of processes forbids reasoning across from the one to the other.

The size of the particles in a flotation pulp affects the percentage of solids and the amount of oil necessary. It is not unlikely, also, that it has some effect on the necessity for other agents. If the solids are coarse it is necessary to run with a thick pulp in order to attain a good recovery. A thick pulp, in

general, results in a low-grade concentrate. Hence a coarse feed is likely to mean a low-grade concentrate. More oil is, in general, necessary, if the feed is coarse. This is probably due to the fact that, owing to the lesser covering power of the coarse material, more of the stabilization of the froth must be done by the oil. The necessity for flocculation of very fine material is not present in the case of coarse feed. Hence the necessity of an electrolyte to produce such flocculation is lacking and the conclusion follows that a coarsely-ground pulp from a given ore is less likely to require the use of acid or alkali than a finely-ground pulp from the same ore.

Oil. "Oil" is a generic term which, in flotation terminology, is used to designate the organic substance that is used to produce frothing and to effect selection of the metalliferous mineral. The "oil" employed is usually an oily substance, but it may be a non-oleaginous organic substance. Rarely, as in the Potter-Delprat process and in certain applications of the agitation-froth and pneumatic processes an inorganic compound may take the place of the "oil."

The character or kind of oil used depends upon: (1) the character of the ore; (2) the fineness to which it is ground; (3) the percentage of solids in the pulp; (4) the method of treatment of the pulp following the introduction of the oil.

The purpose of the oil in froth-flotation is (1) to form, together with the water and solid of the pulp and the gas introduced into the pulp, a froth; and (2), to aid in the selection of the particles of mineral of metallic, resinous or adamantine luster in the pulp from the gangue minerals. Not all oils will perform both of these functions with all ores in all processes. Newly refined paraffin hydrocarbons, if pure, will not froth to a sufficient extent to make them efficient flotation agents in the agitation-froth or pneumatic processes. Certain other substances, although possessing the property of froth formation in these processes, exclude practically all solid matter from the froth. Saponin is such a substance. Certain other agents, such as soap, cause the formation of a froth containing solid matter, but this froth results in no useful concentration. Finally, a considerable

number of oily substances such as essential oils and coal-tars and wood-tars and their fractions and derivatives cause not only copious frothing but, with certain ores, efficient selection of metalliferous mineral from gangue. With other ores the selection is nil or wholly inefficient. It may be put down as an axiom of the art that no one substance is universally applicable as an "oil" in froth-flotation concentration of all ores.

The fineness to which the ore is ground has a considerable effect on the kind of oil necessary. A certain degree of stability is essential in every froth, enough to allow for the removal of the froth from the flotation machine. Stability of froth is affected by the extent to which the bubble walls are covered by solid matter and by the strength of the liquid films of the bubbles themselves. The more closely particles of solid are packed together in the bubble walls and the thinner the layer of liquid between two adjacent pieces of solid, the stronger will be the bubble film. In order to get the greatest covering power from a given lot of ore, it should be ground as finely as possible. With such finely-ground ore little or no aid in stabilizing the bubble films is needed from the oil. If, however, the ore is not so finely pulverized and its covering power is thereby decreased, some agent capable of adding stiffness to the froth must be used, in order to obtain the necessary stability. Pine-tar oil and some petroleum oils are usually used for this purpose.

The percentage of solids in the pulp affects the character of oil necessary in a way similar to that in which it is affected by the degree of pulverization of the ore. A low percentage of solids in a pulp ground to a given degree of fineness means a greater distance between individual particles. It will result from this greater spacing of particles that a bubble will arrive at the surface less heavily loaded than in a thick pulp and will be, therefore, less stable. Such being the case, it will require the addition of an oil with froth-stabilizing power to make up for the lack of stability. Another factor enters here also. If a given amount of solid matter is to be passed through a given machine in a given interval of time, say 24 hours, with a thin pulp a greater volume must pass through in each unit of time

than with a thick pulp. This means that less time is afforded for the dispersion of the oil through each unit of volume of the thin pulp. The rate at which an agent can be dispersed is a specific function of the agent itself, depending principally on its viscosity and its solubility in water, and is also a more or less direct function of the agitation. With the degree of agitation and the rate of solid feed fixed, it follows that with a thin pulp a more mobile or more easily soluble oil, or both, will be necessary than with a thick pulp.

The method of treatment of the pulp following the introduction of the oil determines also the kind of oil most suitable. This follows from a consideration of dispersion. If the pulp is to be subjected to a considerable amount of agitation, either violent or of long duration, or both, after the introduction of the oil and before flotation is to be attempted, then a viscous or only slightly soluble oil will be effective. If, on the other hand, flotation is to be attempted with but little agitation intervening following the introduction of the oil, a mobile and relatively soluble oil must be used.

The quantity of oil necessary in any froth-flotation operation depends upon: (1) the kind of oil used; (2) the recoverable mineral content of the ore; (3) the percentage of solids in the pulp; (4) the degree of pulverization to which the ore has been subjected; (5) the treatment to which the pulp is subjected in the interval between the addition of the oil and flotation; (6) the duration of the flotation operation; (7) the mineralogical composition of the ore; (8) the method of flotation employed.

Mobile and highly soluble oils can be employed in smaller quantity, all other conditions being equal, than viscous and relatively insoluble oil. This follows naturally from the preceding discussion. Mobile and highly soluble oils are easily dispersed in an extremely high state of subdivision, while viscous and slightly soluble oils are dispersed more slowly and to no such high degree. In pulp-body concentration the function of the oil is to coat the mineral particles. In bubble-column processes it is essential that the rising bubbles become oiled. An extremely thin film is all that is necessary. But in order to insure that the

sulphide particles in the one case and the air bubbles in the other shall come into contact with oil, a certain minimum spatial distribution of the oil in the pulp is necessary. In order to insure this minimum spatial distribution with a viscous and relatively insoluble oil, necessarily in relatively large masses as compared with the particles of a mobile and highly soluble oil, a greater amount of the former must be used. Owing to the greater size of the masses of the viscous and insoluble oil the films on the particles and the bubbles will exceed the effective minimum, and further the amount of excess oil which does no coating but which is necessarily present in order to accomplish the required spatial relation will, in this case, exceed in bulk that unused in the case of the mobile or highly soluble agent.

Above a certain minimum quantity, all other conditions being constant, the amount of oil necessary in the agitation-froth process varies directly with the amount of recoverable mineral in the ore. This is easily proven experimentally and can be predicted from theoretical considerations as follows: The maximum surface that can be covered by a given quantity of a given oily substance is measured by the area of the film, one molecule thick, which can be obtained from the given amount of agent. In any successful agitation-froth flotation operation it is essential that all of the sulphide mineral particles be coated to at least this extent. This coating cannot be accomplished without the presence of an excess of the agent in the pulp. Hence the minimum quantity of agent necessary is some probably fixed excess over that required to coat the sulphide particles with a layer one molecule deep, which excess depends upon the degree and duration of agitation, the kind of agent and the thickness of the pulp. Any increase in the amount of metallic mineral in the pulp means an increase in the area to be covered by the oil and hence an increase in the amount of oil that must be provided.

Above a certain minimum quantity, all other things being constant, the amount of oil necessary in the agitation-froth process to make a given recovery from a given ore with a given grade of concentrate varies directly with the percentage of moisture in the pulp, within the efficient working range of

moisture percentages which is from, say, 65 or 70 per cent. to 90 or 95 per cent. This is confirmed by experimental data and follows logically from a theoretical analysis. As has been previously stated, a certain minimum spatial distribution of the particles of oil in the pulp is necessary in order that the metalliferous mineral particles may be coated during the time that the pulp is under treatment. If the volume of pulp carrying a given amount of solid matter is increased, then the number of particles of oil necessary to produce the minimum spatial distribution of the same throughout the total volume of pulp will likewise be increased.

The following relations between quantity of oil and size to which the ore is ground are experimentally proven: (1) If a pulp containing solid matter ground to a given degree of fineness is being concentrated by flotation with a given minimum quantity of a given agent, the same metallurgical results can be obtained with a smaller quantity of agent, if the solids are more finely ground. Conversely more oil must be used, if the grinding is so changed that the product to be floated is coarser. The explanation of this observed phenomenon is, probably, that a certain degree of stability is essential in the froth and that this stability may be provided by either oil or solid matter. If the covering and hence stabilizing power of the solid is increased by finer subdivision, the oil is relieved of part of its duty and less of it, therefore, is necessary. *Vice versa*, if the covering and stabilizing power of the solid is decreased, as by coarser grinding, more burden is placed on the oil and it must be increased in quantity.

The treatment to which the pulp is subjected in the interval between the addition of the oil and the actual operation of flotation has a considerable influence on the quantity of agent used. A certain minimum degree of dispersion of the agent is essential, as has been previously explained, in order to assure such spatial distribution of the agent that every air bubble or sulphide particle shall become coated. It is necessary further that the agent be dispersed to such a degree that gravity will have little or no effect to cause it to collect together and that the tendency of the particles of the agent to coalesce shall be overcome.

Otherwise the agent will collect on the surface of the pulp and break down the froth. Hence, if an oil of a given character is employed, the amount of agitation required to disperse it must be increased, as the amount of oily agent is either decreased or increased from a certain amount corresponding to a minimum degree of agitation for mixing.

The quantity of oil necessary and the duration of the flotation operation, all other conditions being constant, bear a close relation to each other. With any given minimum amount of agent and minimum duration of the flotation operation corresponding thereto required to attain a certain result, a lessening of the duration of the operation will require an increase in the amount of oil and *vice versa.*

The mineralogical composition of the ore, particularly as regards the gangue, affects the quantity of oil necessary. Clean, hard ores require less oil, all other conditions being constant, than do soft clayey ores. One reason for this difference probably lies in the fact that the clayey gangue matter from the soft ores acts to emulsify oil and thus prevents a certain amount of the latter from playing any part in the process. Some sulphides, also, require a less quantity of oil to separate them from a given gangue than do others. Thus galena can be separated from a given ore with less oil than can blende, and chalcocite is separated with the use of a less quantity of oil than pyrite. Some methods of differential flotation are founded on this phenomenon.

Pulp-body-concentration processes in general require more oil to be employed, all other things being equal, than do bubble-column methods. This is due to the difference in the phenomena acting in the two cases. In the pulp-body processes using oil it is necessary that the sulphide particles become coated in the pulp before effective selection takes place and this necessitates thorough and quick dispersal of the oil. Further, a given particle of oil is effective only during the short time that it is below the pulp surface. In the bubble-column processes, where all of the work of the flotation agent is done above the pulp surface, coating of all the bubbles in the pulp is not necessary (since those that rise without coating will be coated in the bubble

column itself) and hence the need for so large a number of oil globules floating around therein does not exist. Further, each oil-coated bubble serves to separate a considerably greater bulk of solid than is done by a corresponding area of bubble surface in the pulp-body processes. It should be noted also in this connection that, on account of the much larger size of the bubbles in the bubble-column processes, a bubble rising through a pulp containing particles of flotation agent in a given spatial distribution, is much more certain to meet such particles than the very much smaller bubble in the agitation process.

Minor Agents. The rôle of the minor agents is to increase the grade of concentrate, i.e., aid in selection, and to a lesser extent, aid recovery. Various theories have been advanced to explain their action. In general they are electrolytes, and ingenious hypotheses have been based on assumed accentuation, due to their ions, in the difference in magnitude of the electrical charges said to exist at the surfaces of the solid particles in the pulps. Excluding for the present the cases in which the minor agent reacts chemically with the principal agent or oil, it is a commonly observed experimental fact that successful use of a minor agent is accompanied by increased flocculation of the flotation pulp, particularly of the flotation tailing. It is furthermore usually true that the tailing from an unsuccessful flotation operation is slow-settling, indicating a lack of flocculation. Hence we may set down as an empirical rule that a suitable minor agent will be one that flocculates the pulp.

When the minor agent reacts chemically with the principal agent, the cause to which its effect is to be ascribed is masked. It is not improbable that in such cases any improvement is as much due to the changed character of the principal agent as to the independent flocculating effect of the minor agent. Particularly is this true when the addition of the minor agent results in increased recovery accompanying increased frothing. Such chemical reaction is to be looked for where the principal agent is a vegetable or animal oil, and the minor agent an alkali.

The necessity for the use of a minor agent depends upon, (1) the mineralogical composition of the ore, and (2) the char-

acter of principal agent employed. If the ore is unaltered, hard and silicious, it is certain that with some principal agents no minor agent will be necessary. With other principal agents, however, the grade of concentrate may be improved by the use of a minor agent. If the ore is much altered with a resultant large amount of kaolinized gangue, a minor agent is likely to be necessary, although here, also, the character of the principal agent may be such that the use of a minor agent can be dispensed with.

The minor agent most widely used is sulphuric acid. Sodium hydroxide, sodium carbonate, sodium sulphate, sodium silicate and lime are not uncommonly employed. It is the writer's experience that so far as flotation itself is concerned, when a minor agent is needed, sulphuric acid is the cheapest and most effective. Its use may, however, as is the case with partially oxidized copper ores, be prohibited on account of the destructive effect of the resultant copper sulphate solution on the iron with which the pulp comes in contact in the mill, in which case an alkaline or neutral electrolyte may be effective.

Percentage of solids affects kind and quantity of principal flotation agent necessary, the quantity of minor flotation agent, the fineness of grinding, the grade of concentrate and the recovery attained. It is determined principally by the character of the ore. The effect on principal flotation agent and fineness of grinding has already been discussed.

The relation between percentage of solids and quantity of minor flotation agent seems to point to the conclusion that it is the concentration of the aqueous solution of the minor agent, rather than the quantitative relation between the minor agent and the solid, which determines its effectiveness. No systematic experimental work along these lines is reported but mill experience and laboratory experiment both indicate that the amount of minor agent necessary increases with decrease in the percentage of solids.

Grade of concentrate is, in general, improved by decrease in the percentage of solids in the pulp, and such improvement may be accompanied by improvement in recovery, but continued

decrease in percentage of solids eventually results in decrease in recovery.

The character of the ore determines the maximum economic percentage of solids. With clean, sandy ores this is about 30 per cent., with clayey or slimy ores, about 15 per cent. Slimy pulps as thin as 5 per cent. solids can be treated, although generally at the expense of a low recovery.

Temperature. Heat aids gas precipitation from the water of the flotation pulp; it also aids in dispersion of the more viscous oils. Temperatures above normal are, therefore, of distinct advantage in pulp-body concentration. Heating can be dispensed with, however, by increasing some other factor such as the concentration of gas in chemical-generation processes, the duration or intensity of agitation in the agitation-froth process or the pressure difference in pressure-relief processes. In bubble-column processes, where bubble precipitation from solution is not the essential phenomenon in furnishing the effective gas, heat is not one of the important factors, and the temperature of the pulp is, therefore, unimportant, except insofar as heat may be desirable to aid dispersion of the oil.

In the agitation-froth process, it is, in general, more economical to attain enhanced bubble precipitation by greater intensity and duration of agitation than by heating the pulp, and in bubble-column processes difficulties in dispersion of the oil are surmounted by adding the oil to the grinding mills, or using an oil easy to disperse. Heating is, therefore, rarely resorted to in present-day practice.

CHAPTER II

TESTING LABORATORY EQUIPMENT

Introductory. The apparatus listed in the following pages comprises the equipment necessary for complete and thorough testing of flotation processes. Some of it can be omitted where the problems to be studied are of a special character. It is urged, however, that such omission be of the actual flotation machines themselves rather than in the apparatus listed for preliminary tests. This is something that need not be demonstrated to the man who is experienced in flotation testing. Such a man knows that time spent in determining fully the mineralogical character of the ore to be dealt with will be time saved when the difficulties of actual flotation testing arise.

Preliminary Examination. Determination of mineralogical composition will require a part or all of the following equipment:

Books: "Introduction to the Study of Minerals," Austin Flint Rogers, McGraw-Hill Book Co., Inc., New York; Hill Publishing Co., Ltd., London. "Minerals in Rock Sections," Lea McIlvaine Luquer, D. Van Nostrand Co., New York. "Elementary Chemical Microscopy," Emile Monnon Chamot, John Wiley and Sons, Inc., New York; Chapman and Hall, Ltd., London. "Microscopical Determination of the Opaque Minerals," Jos. Murdock, John Wiley and Sons, Inc. "Microscopic Examination of the Ore Minerals," W. Myron Davy and C. Mason Farnham, McGraw-Hill Book Co., Inc. Apparatus and reagents for blowpipe analysis. A list is given by Rogers. Apparatus and reagents for qualitative microchemical analysis, including a simple polarizing microscope fitted with vertical illuminator for use in examinations by reflected light and with an eyepiece micrometer. Chamot describes such a microscope and lists such other supplies

as are necessary. Apparatus and reagents for preparing rock sections and polished specimens for optical examination. Luquer, Murdock, and Davy and Farnham give complete lists. In connection with this latter apparatus, if but few rock sections are to be prepared and examined, the mechanical polishing apparatus can be dispensed with. Preliminary examination will be much facilitated by the use of a binocular microscope equipped with 25-mm. and 40-mm. objectives and 6X and 10X oculars. This microscope should have the usual stand with stage for work with transmitted light and also a small stand without stage which allows the microscope to be brought to and placed upon large objects, making possible a study of the same, without the necessity of preparing small specimens. Such a microscope as this is essential for investigation of the behavior of flotation froths and pulps in later stages of the testing work. An ordinary enameled gold pan and vanning plaque will be of aid also in the preliminary work and are necessary in later work.

Preparation of the ore for flotation will necessitate the use of the following equipment: Set of sizing screens. The Tyler standard screen scale sieves, ranging in size of aperture from 2.362 mm. to 0.074 mm., are suggested. Pair of platform scales with 500 lbs. capacity for weighing in lots of ore. Small sample jaw crusher. Braun disk pulverizer. Laboratory ball mill about 18 ins. by 18 ins. Set of balances of the druggist trip scale variety, 10-lb. or 5-kilogram capacity, with beam graduated to 500 grams by 10-gram intervals and to 16-oz. by $\frac{1}{4}$-oz. intervals. Six-inch by six-inch Jones riffle complete with four pans and scoop. One-pound paper sacks for sacking samples preliminary to testing. Supply of 1-qt. and 2-qt. Mason jars for wet pulp samples.

Flotation Testing

Machines. The machines described below are more than will be needed in an ordinary flotation testing laboratory, but the descriptions are included here for the sake of completeness.

The particular field of each machine is stated in connection with the detailed description.

Gabbett mixer. This piece of apparatus is illustrated in U. S. patent 835,120 as having been used in the treatment of ores by agitation to produce a froth concentrate, and is the best laboratory apparatus in which to make a froth concentrate of the nature described in that patent. If a study is to be made of the development of froth flotation processes, this machine should be in the laboratory. The laboratory machine is shown in Fig. 1. It consists of a glass cylinder (*a*) open at both ends, mounted vertically in a hemispherical bronze casting (*b*). This casting is fitted with outlets (*c*) and (*d*) and is mounted on legs (*e*). These are fastened to a plank which forms the base of the stand carrying the agitating mechanism. The agitating mechanism consists of a stirrer (*g*) which has the shape of the frustum of a cone and is usually placed as shown with the large end down. This conical shell is fastened to the vertical spindle (*h*) by means of arms (*i*) and (*j*). The vertical spindle is supported by the post bearing (*k*) with two boxes (*l*) and (*m*) respectively and the thrust bearing (*n*) at such a height that the lower edge of the conical agitator clears the bottom of the chamber by but $\frac{1}{4}$ to $\frac{1}{2}$ in. The shaft is driven by means of a grooved pulley (*o*), 2-in. diameter, and a $\frac{3}{8}$-in. round leather belt from the motor. Removable baffles (*s*) are provided for use when a high degree of agitation is desired. The post bearing (*k*) is carried on the vertical member (*p*) fastened by an angle to the base and braced. An annular overflow launder (*r*) is fitted to the cylinder (*a*). A $\frac{1}{4}$-h.p. variable speed horizontal motor should be used to drive the machine and should be set up with its pulley not over 1 ft. from the center of the driven pulley, if belt trouble is to be avoided. The motor speed and pulley size should be such that the speed range of the mixer spindle will be from 200 to 2000 r.p.m. A machine of the size shown takes a charge of 300 to 350 gms. of ore according to the percentage of solids desired in the pulp.

The Slide Machine was invented to overcome the difficulty encountered in removing froth from the Gabbett machine. It

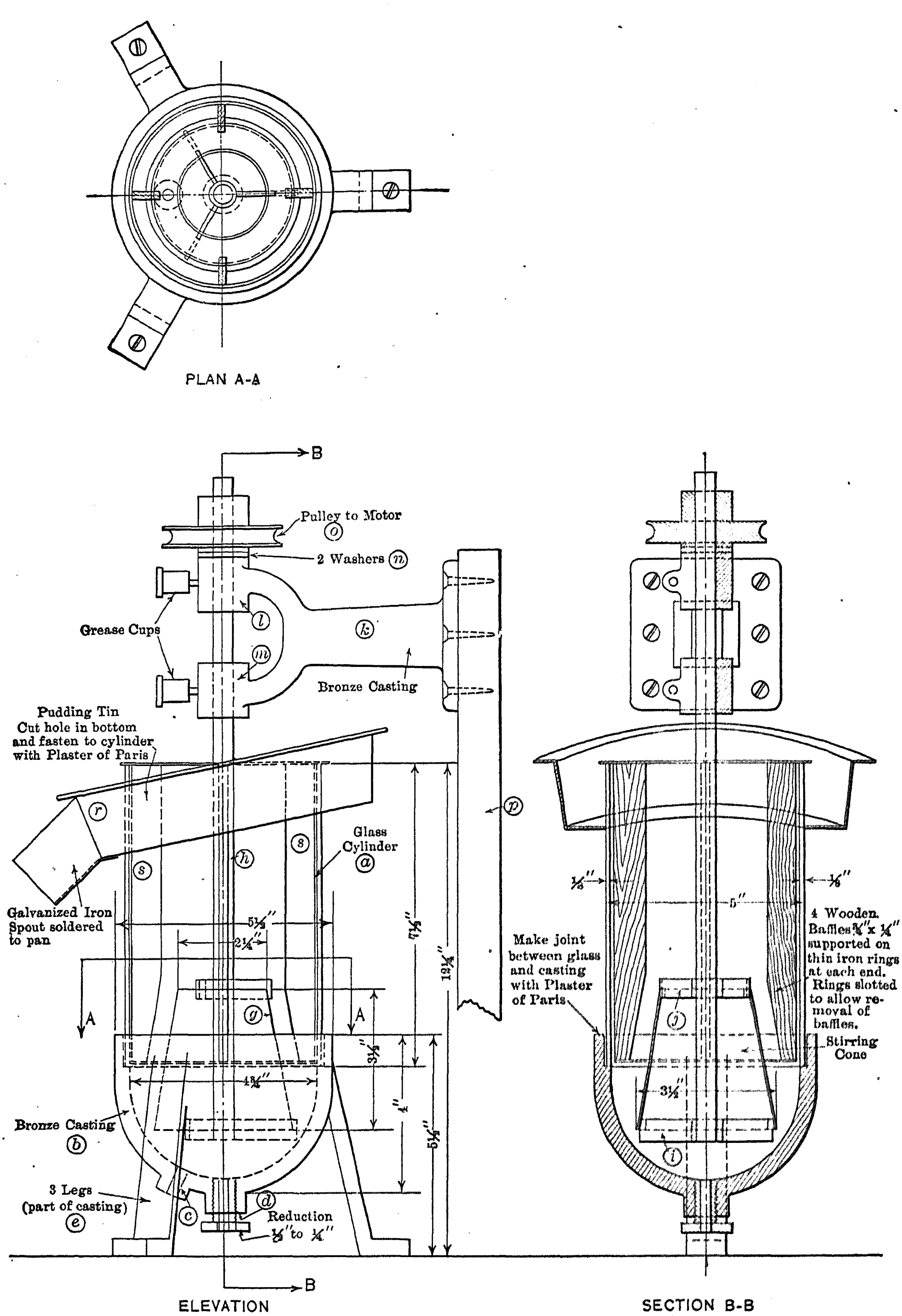


FIG. 1
Laboratory Gabbett mixer

is shown in Fig. 2 and consists of a square box (*a*) supported on four legs as shown. The top of the box carries a trough (*b*) with the bottom flush with the upper edge of the box and with sides

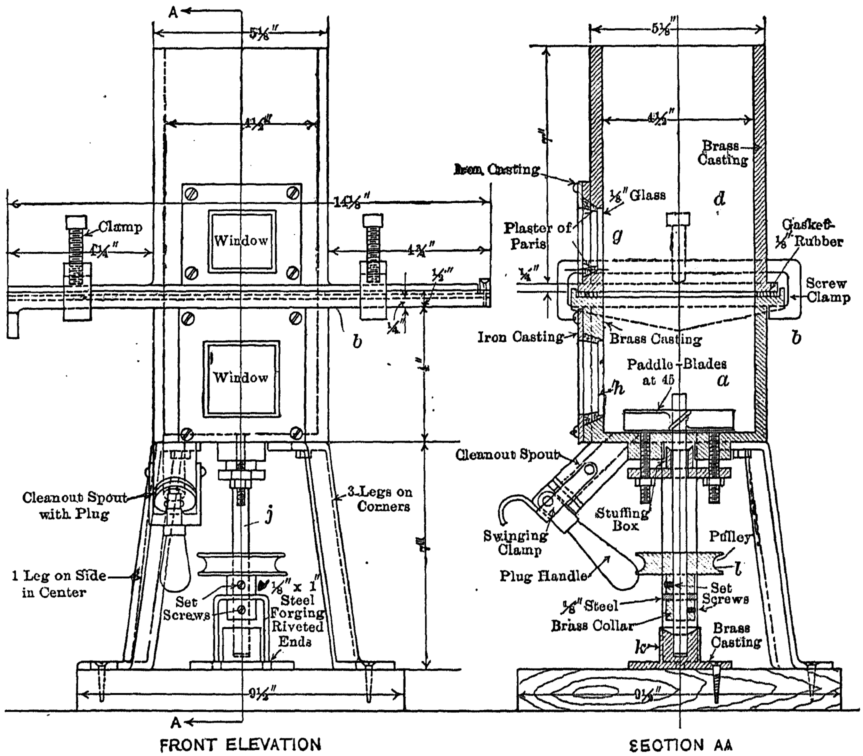


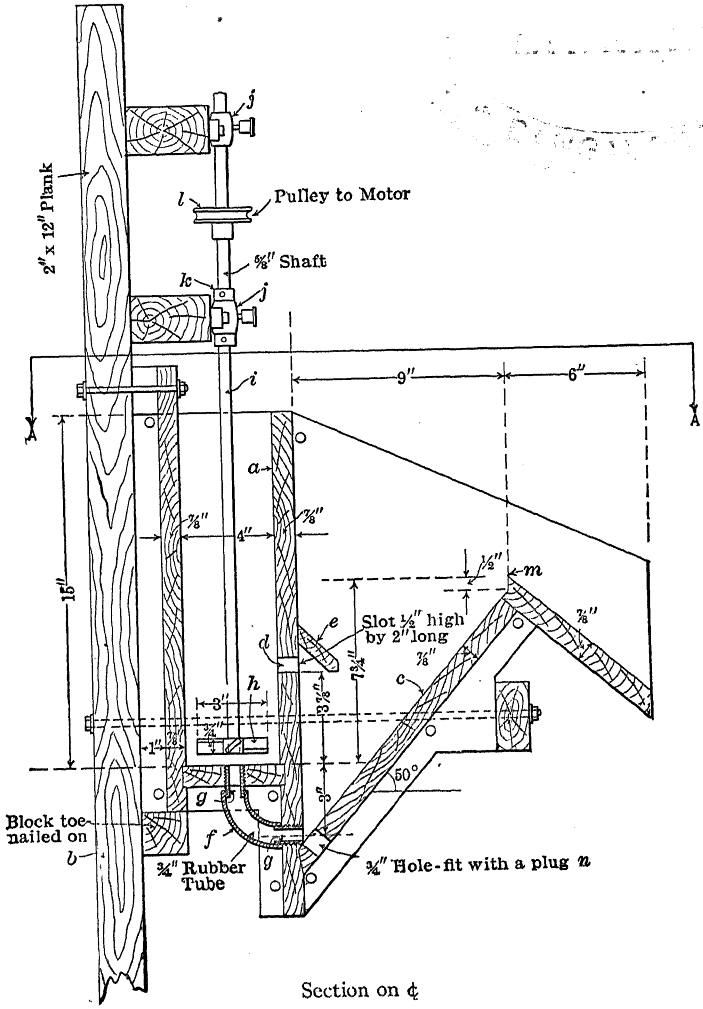
FIG. 2
Laboratory slide flotation machine

about $\frac{1}{4}$ in. high. Above the box (*a*) is placed the sliding square pipe (*d*) of the same cross-section as the box and flanged to fit loosely in the trough (*b*) and to slide therein. A rubber gasket is provided which fits the surface of the trough and permits of a tight joint between the upper and lower sections of the machine when the two are clamped together by means of screw clamps. Both box (*a*) and slide (*d*) are provided with plate glass windows (*g*) and (*h*). Agitation is effected by means of a four-armed stirrer carried on the vertical spindle (*j*) which passes

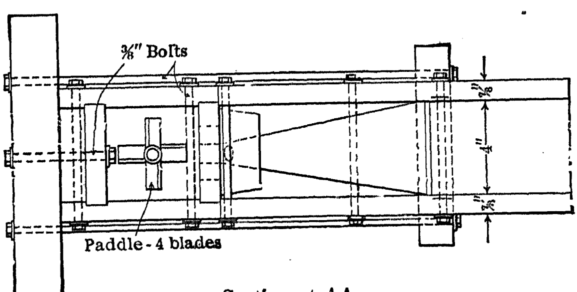
through a stuffing box in the bottom of the box (*a*) and is carried on the step bearing (*k*). This shaft is actuated by a grooved pulley (*l*) and round leather belt from a $\frac{1}{4}$-h.p. horizontal motor. This motor should give a speed range of from 800 to 2400 r.p.m. and should be set up with not more than a foot between pulley centers. The charge for this machine is 300 gms. of ore.

For ordinary testing of the agitation-froth flotation process this machine has been superseded and it now has no place in any but the most complete flotation laboratory.

Minerals Separation Standard machine. A laboratory model of this machine is manufactured and sold by the Denver Fire Clay Co., under the name of the Case laboratory flotation machine. It can, however, be made up in the laboratory for much less money than it can be purchased, and the home-made machine will give just as satisfactory results. Such a machine is shown in Fig. 3. It consists of a square box (*a*) made of $\frac{7}{8}$-in. plank, or, better, metal, mounted on a plank (*b*). At one side of the agitating compartment (*a*) is joined a froth-separating compartment (*c*). Pulp passes from the agitating compartment into the froth-separating compartment through the slot (*d*) over which is placed the baffle (*e*), the purpose of which is to lessen the disturbance of the surface of the pulp in the froth-separating compartment. Circulation of pulp is accomplished through the pipe (*f*) which consists of a rubber hose slipping over nipples (*g*) in the back of the froth-separating compartment at the bottom and in the center of the bottom of the agitating compartment. Agitation is effected by means of a four-armed stirrer (*h*) attached to the bottom of a vertical spindle (*i*) which latter is supported by the bearings (*j*) and the thrust bearing (*k*) and is actuated through the grooved pulley (*l*) and a quarter-turn round belt from a $\frac{1}{4}$-h.p. motor, placed best not more than 1 ft. distant. The froth-separating compartment (*c*) is fitted with an overflow lip (*m*). Tailing is withdrawn from the machine by removing the plug (*n*). The motor should be variable speed so connected as to allow a speed range for the vertical spindle of from 800 to 2000 r.p.m. The machine shown takes a charge of 750 gms. of ore.



Section on ℄



Section at AA

FIG. 3

Minerals Separation laboratory flotation machine

This type of machine is not the most satisfactory for laboratory testing for the reasons that the froth-separating compartment is too large, the pulp circulation is poor, and it is difficult to clean up. It can be improved by narrowing the froth-separating compartment. For routine tests of the agitation-froth process the Janney laboratory machine is far superior.

The Janney flotation test machine shown in Fig. 4 consists of a cylindrical agitation compartment (*a*), fitted on the periphery with radial baffles (*b*). Attached to one side of the agitating compartment is a froth-separating compartment (*c*). The front of this froth-separating compartment is cut down to a beveled edge to allow overflow of froth. Around the agitating compartment at the top is placed the annular launder (*d*). The machine is covered with a removable hemispherical cover (*e*) with a hole at the top. The base of the casting forming the agitating compartment is flanged and sits upon a frame (*f*). Agitation is effected by means of two four-armed impellers placed one near the bottom and one near the top of the agitating compartment. These impellers are carried on the vertical spindle (*g*) which passes through a stuffing box in the bottom of the agitating compartment. The lower end of this spindle is carried in a step bearing (*h*). The vertical shaft is actuated by means of a quarter-turn round leather belt and a grooved pulley (*i*) from a $\frac{1}{4}$-h.p. variable speed motor set preferably with pulley not more than 1 ft. away. The agitating compartment is fitted with a spout (*j*) and brass plug with handle (*k*) for draining residues from the machine. The motor should be such as to give a speed range of from 500 to 2400 r.p.m. A useful addition to the above machine consists in an upper removable bearing for the agitator shaft. This is shown in the figure and consists of a bearing (*l*) carried on the horizontal pieces (*m*) which are fitted to slide freely on the uprights (*n*) and are stopped by the collars (*o*). The vertical spindle is now extended to pass through the bearing (*l*), thus doing away with whipping of the shaft during agitation and lessening materially the wear on the stuffing box and step bearing. This shaft extension also makes possible the determination of the speed of the impeller shaft, which is

impossible in the machine as sold. The ore charge to produce a pulp of 20 to 25 per cent. solids is 500 to 600 gms. The Janney machine is the best apparatus for routine testing by the agitation-froth process. It may be purchased from the Stimpson Equipment Co., Salt Lake City, Utah.

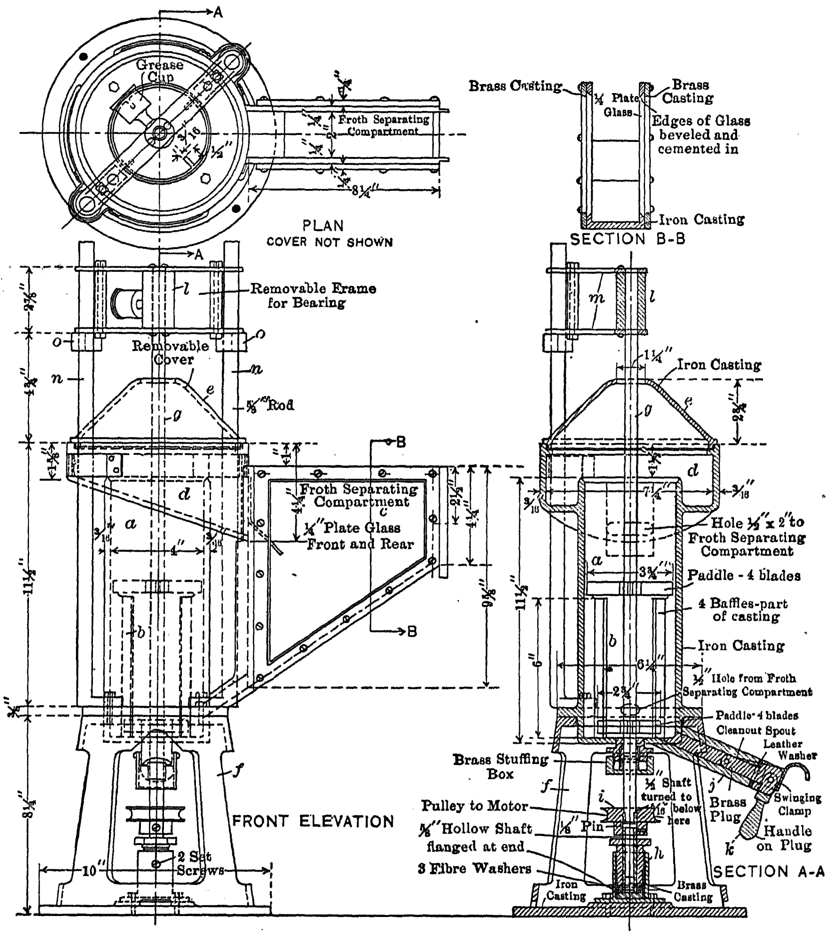


FIG. 4
Janney laboratory machine

The Elmore vacuum laboratory machine. A convenient form of laboratory apparatus for operating the Elmore vacuum

process is shown in Fig. 5. In the assembly sketch, (*B*) is the feed funnel discharging through the valve or stop cock (*A*) and the tube (*H*) through the bottom of the plate (*I*) into the

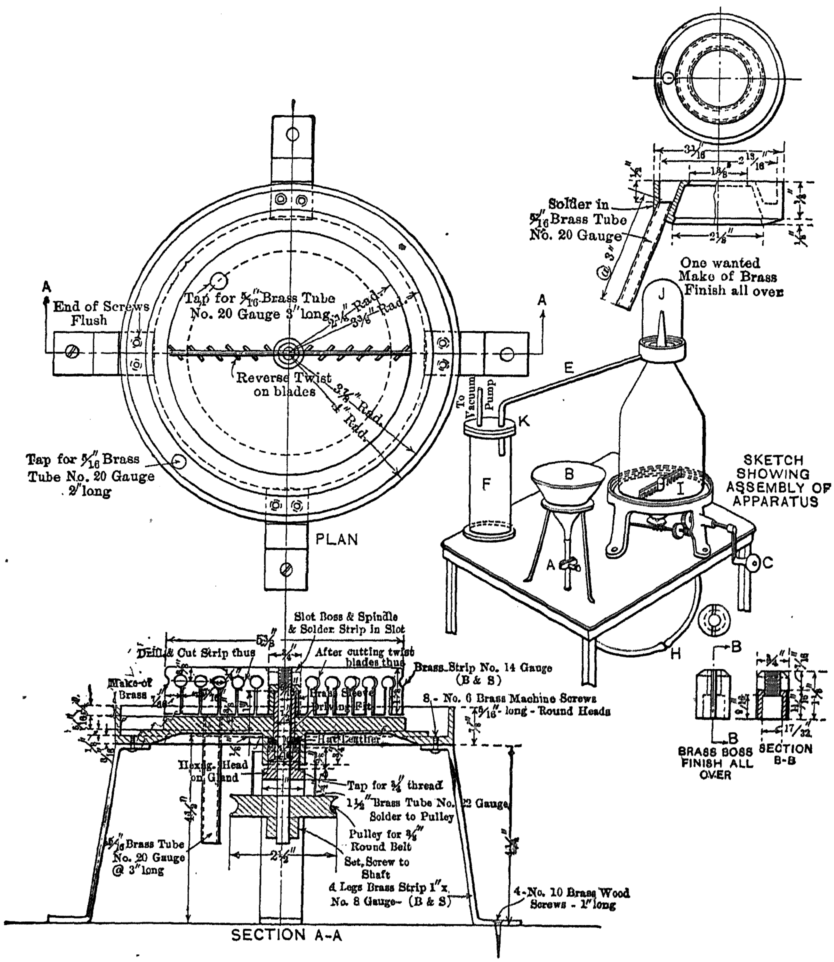


FIG. 5
Elmore vacuum laboratory machine

vacuum chamber. Suction is applied to the pulp in the vacuum chamber through the dome (*J*), pipe (*E*), and concentrate receiver (*F*), by means of a pipe connecting the latter with the

vacuum pump. Operation of the apparatus is intermittent. After a full charge is fed in, the valve (*A*) is closed, suction is applied and the rake slowly revolved. The froth which rises to the surface passes up through the inverted funnel into the dome (*J*) and out through the pipe (*E*) into the bottom of the concentrate collecting chamber (*F*). After all froth has been taken off, the vacuum is relieved and tailing withdrawn through the pipe shown at the left hand side of section (*AA*) passing through the bottom of the plate (*I*). It will be noted that the detail drawings show provision for belt drive for the rake rather than for the bevel gear drive illustrated in the assembly. The belt drive is simpler and cheaper. The funnel shaped cover for the vacuum chamber and the dome (*J*) are made of glass about $\frac{8}{32}$ inch to $\frac{1}{8}$ inch thick. The concentrate receiver (*F*) can well be made a wide-mouthed bottle. It is planned to operate the rake at 10 to 12 r.p.m. The machine takes a charge of 300 to 500 gms. of ore.

This machine is extremely satisfactory for laboratory testing of ores by the vacuum process. Good recoveries and high grade concentrate can be made and the phenomena are clearly visible.

The K and K laboratory flotation machine shown in Fig. 6 is manufactured by the Braun Corporation, Los Angeles, Cal. It consists of a horizontal split cylinder (*a*), the lower half being carried on legs while the upper half is hinged at the back and carries the bearings for the rotor shaft. Cast in one piece with the lower half of the cylinder just described is a froth-separating box (*b*) the length being the same as that of the cylinder. This box connects with the aërating chamber by ports (*c*) through which aërated pulp enters and port (*d*) by means of which pulp passes back for more aëration. It is also fitted with a drain cock (*e*) for drawing off the residual pulp after treatment. The aërating mechanism consists of a slatted cylinder made up by fastening slats (*f*) to the large slats (*g*) which are in turn bolted to spiders (*h*) carried on the shaft. The shaft is driven by pulley as shown. The machine will require a $\frac{1}{4}$-h.p. variable speed motor fitted with a crown pulley to take a 2-in. belt. Motor pulley and speed should be such that with a 6-in. pulley

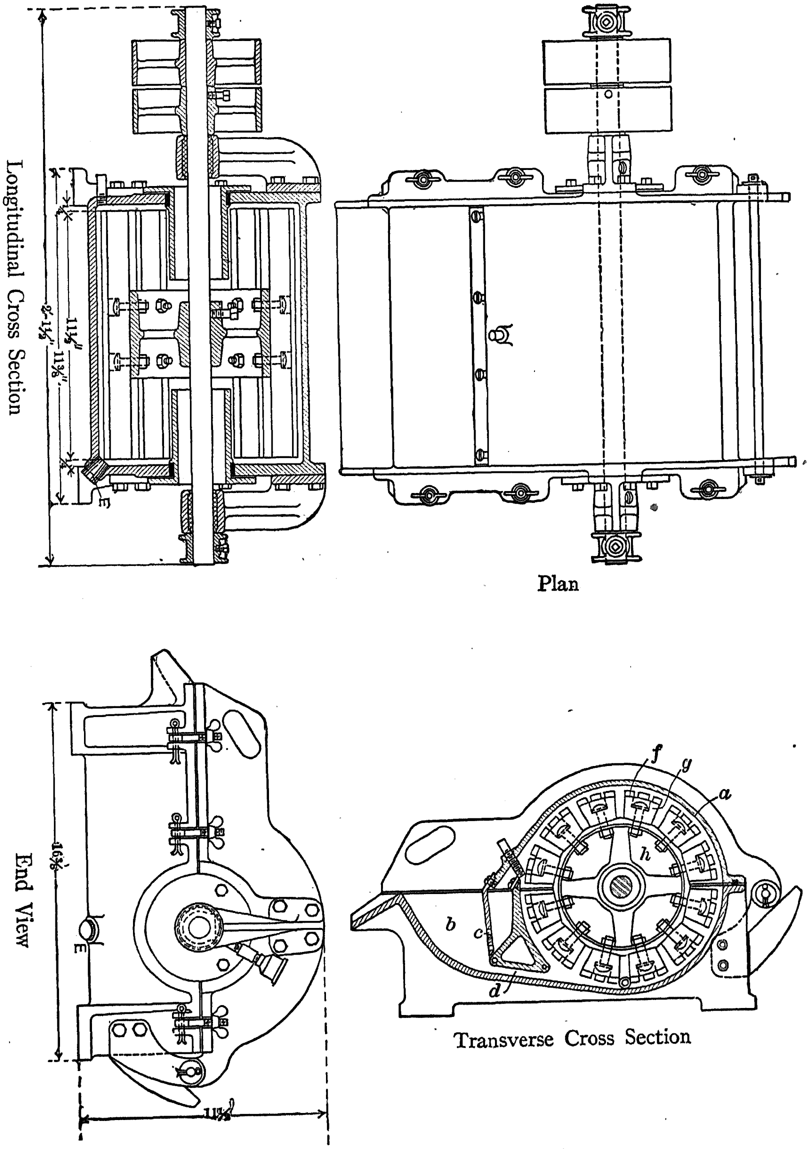


FIG. 6
K & K laboratory flotation machine

on the machine, a speed range of from 200 to 800 r.p.m. can be attained. This machine requires a charge of about 750 gms. of ore for a test with a pulp containing 20 to 25 per cent. solids. The machine is useful for studying the behavior of an ore subjected to bubble-column concentration with an amount of air considerably less than that introduced in pneumatic machines.

The Ruth flotation testing machine consists of a box (*a*) partially divided, as shown in Fig. 7, into an aërating compartment (*b*) and a froth-separating compartment (*c*). The froth-separating compartment is connected at the bottom with the aërating compartment by means of a passage (*k*) which connects the bottom of this compartment with the bottom of the aërating compartment, entering directly under the vertical spindle. Aëration is accomplished by means of the hollow disk (*d*) which is carried on the lower end of the hollow vertical shaft (*e*). This latter is supported by post bearings (*f*) and thrust bearings on the frame (*h*). The shaft is actuated by means of a $\frac{3}{8}$-in. round leather belt passing over the grooved pulley (*i*). Another form has two vertical grooved guide pulleys on the frame allowing the motor to be mounted on the same base as the machine. The hollow disk on the lower end of the vertical spindle is shown in plan and section in the figure. Ports (*g*) circulate pulp by centrifugal force. Air enters through the hollow shaft and ports (*l*) to fill the vacua formed behind the shields (*m*) as the disk rotates. A grid (*n*) prevents disturbance of the surface of the pulp and aëration of the pulp from the surface. A $\frac{1}{4}$-h.p. variable speed motor giving a speed range of from 500 to 2000 r.p.m. should be provided. The solid charge required for a machine of this size with a pulp of 20 to 25 per cent. solids is 500 gms. This machine is sold by the Mine and Smelter Supply Co. of Denver, Col. The Denver Engineering Works Company is the manufacturer and sales agent for the mill-sized machine. The Ruth machine is of the bubble-column type but introduces less air per unit volume of pulp than the pneumatic type machines. The testing machine is efficient for this kind of operation.

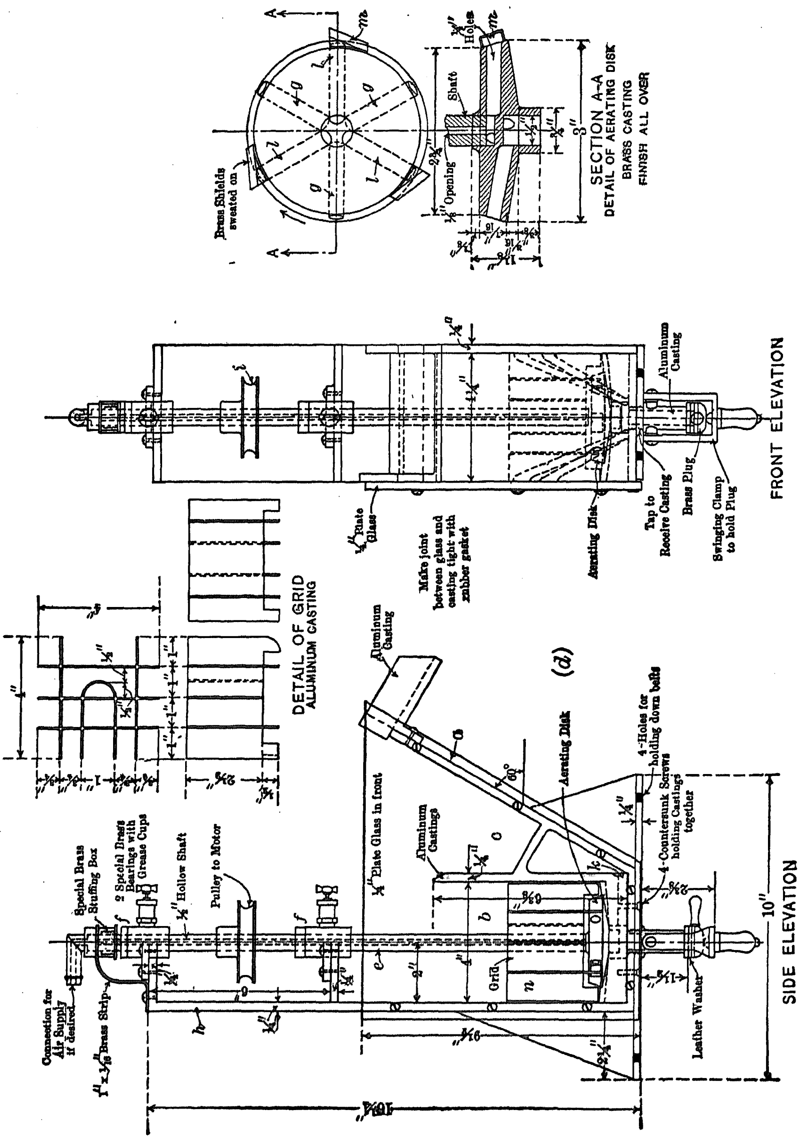


FIG. 7
Ruth laboratory flotation machine

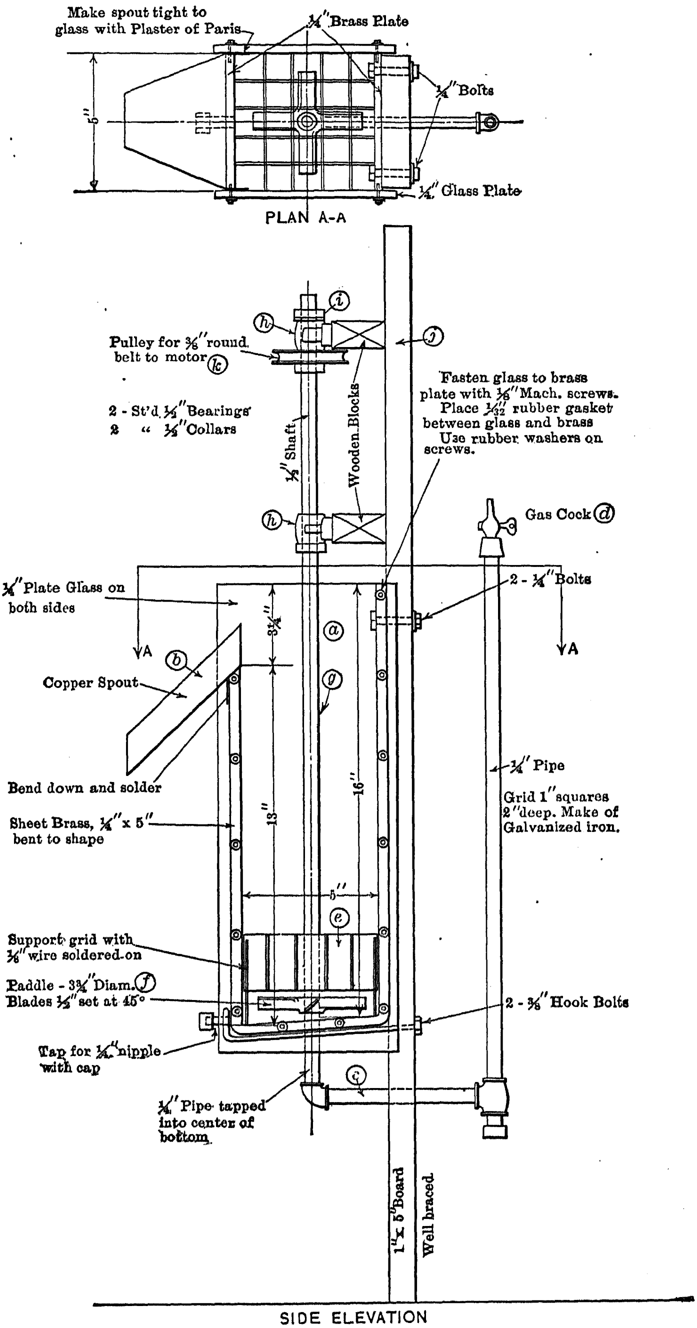


FIG. 8
Laboratory sub-aëration machine

The Hebbard sub-aëration laboratory machine (Fig. 8) consists of a square box (*a*) with slightly sloping bottom and with the front side cut down to allow discharge of froth over the lip (*b*). To the bottom of the box is attached a pipe (*c*) which is carried up outside the machine to a point about 3 ins. above the top and is there terminated by a gas cock (*d*). A grid (*e*) is placed with its lower edge about 1 in. above the bottom to prevent the formation of a vortex at the surface. Aëration is accomplished by means of the four-armed cross-shaped beater (*f*) fitted with arms at 45° from the horizontal and mounted on the lower end of the vertical spindle (*g*). This spindle is carried on the bearings (*h*) and the thrust bearing (*i*) from the framework (*j*) and is driven by means of a $\frac{3}{8}$-in. round quarter-turn belt and the grooved pulley (*k*) from a $\frac{1}{4}$-h.p. variable speed motor, set up at a distance preferably not more than 1 ft. The speed range should be such as to allow variation of from 500 to 2000 r.p.m. of the vertical spindle. This machine requires a charge of 750 gms. of ore for a pulp containing 20 to 25 per cent. solids. The machine introduces much more air than is introduced by the agitation-froth type machine and the concentration obtained is of the bubble-column type. It is a satisfactory piece of laboratory apparatus.

The Callow pneumatic laboratory flotation apparatus furnished by the General Engineering Company, Salt Lake City, is illustrated in Fig. 9. It consists essentially of a rougher cell 2 ins. wide by 15 ins. long and a cleaner cell 2 ins. wide and 7 ins. long, with an air lift for returning products to the head of the rougher cell. The rougher flotation cell itself consists essentially of a rectangular box (*a*), with a porous bottom made of several plies of light weight canvas or palma twill stitched together and clamped between the top of a four-compartmented chamber or air box (*b*) and the flanged bottom of the box (*a*) by means of screw clamps (*c*). Each compartment of the air basket is connected through the valves (*B*) with a header, in order, by adjustment, to permit equal distribution of air against the varying head at different points in the length of the cell. The depth of the cell at the head end is about 6 ins. and at the tail end about

11 ins. The cell is provided with an overflow launder (*d*) on both sides. These launders join at the tail end in a spout which delivers rougher froth to the cleaner cell. The rougher cell is drained by a pipe with a two-way cock (*F*) which discharges through pipes (*G*) and (*H*). Outlet (*G*) discharges into the air lift. It permits regulation of the pulp level in the rougher cell and allows circulation of rougher tailing during a test, if desired. Outlet (*H*) is for draining and washing out the rougher cell at the end of a test. The cleaner cell has a two-compartmented air basket and is fitted with an overflow launder similar to that on the rougher cell. Pulp level in the cleaner cell is regulated by the inclination of the pipe (*E*). The overflow from this pipe goes to the air lift and back into the system. This apparatus requires a charge of 1500 gms. of ore for a test. Eleven cubic feet of air per minute at 5 lbs. pressure should be furnished.

Figure 10 shows a much less elaborate apparatus, which the writer has found to be entirely satisfactory for laboratory testing of pulps by the pneumatic process. It consists of the usual rectangular box (*a*) with sloping porous bottom (*b*) fitted, however, with overflow launder (*c*) on one side only, the other side being of glass to permit observation of the interior. The construction is sufficiently shown in the sketch. The machine can be operated continuously by arranging feed rate and the tailing-discharge slide (*d*) to permit sufficient treatment time, or the tailing discharge pulp can be circulated by means of an air lift. If intermittent non-circulating treatment is desired the tailing gate can be stopped up by suitable means. The charge of ore for a pulp containing 20 to 25 per cent. solids is 1500 gms.

The General Engineering Co. recommends a positive rotary blower of the Root or Connersville type and says that the oil in the air from a compressor chokes the blanket. The writer has found, however, that a small compressor such as the Ingersoll-Rand Imperial Type 14, air-cooled, 2½ ins. by 3 ins., using an ordinary house hot-water tank as a receiver, makes a more satisfactory installation than a blower. Pressure in the receiver should be maintained at about 10 lbs. Pressure on the blanket side of the regulating valves will vary from 1 to 3 lbs.

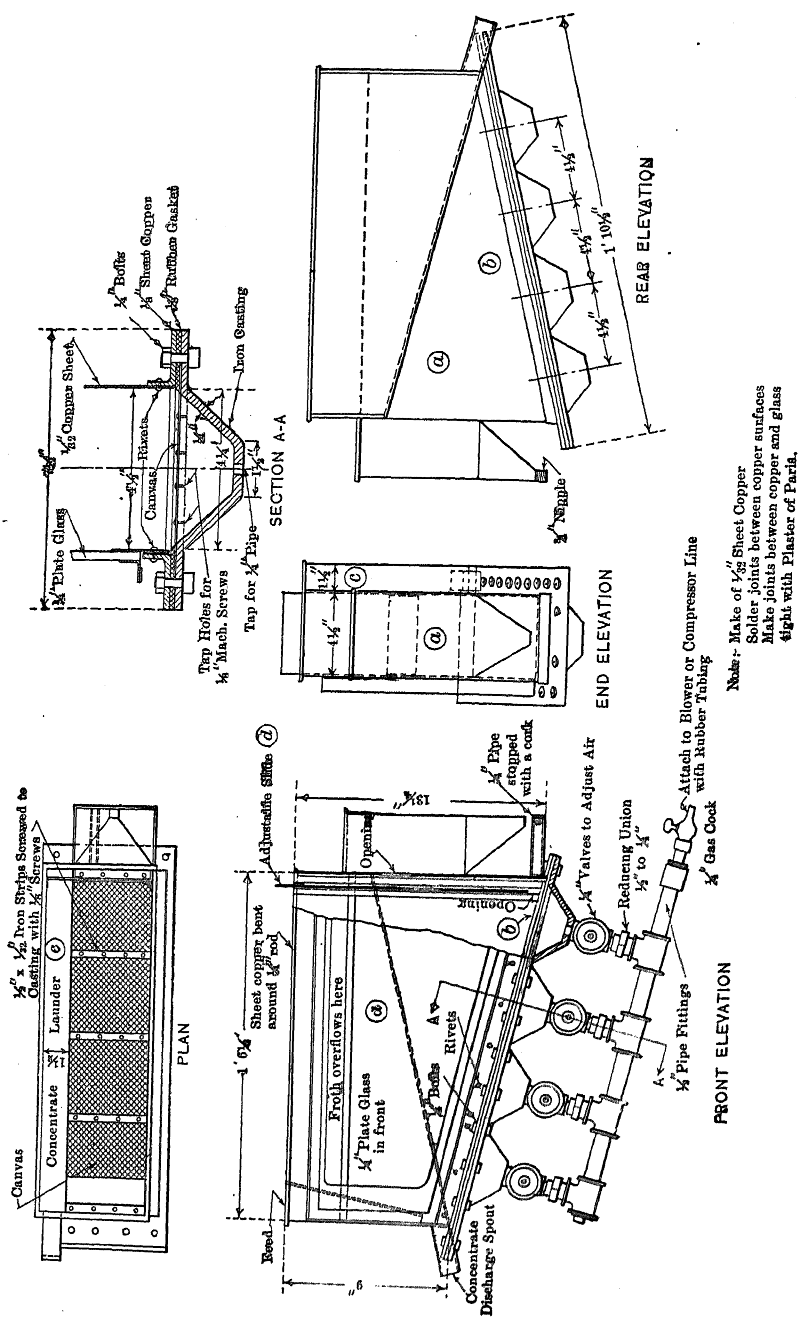
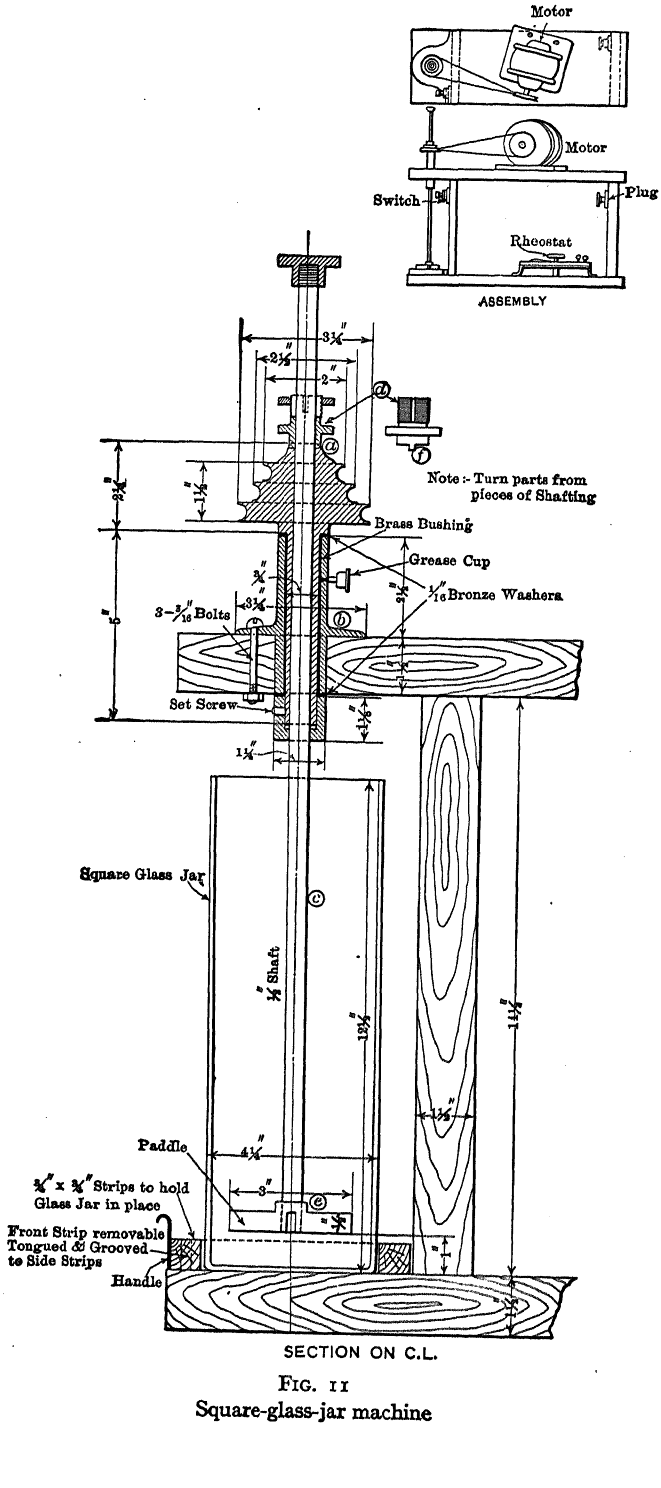


FIG. 10
Laboratory Callow cell



SECTION ON C.L.

FIG. 11
Square-glass-jar machine

A pneumatic machine of one of the types above described should be in every general flotation laboratory.

For preliminary qualitative tests of the agitation-froth process and for all kinds of stirring and mixing operations, a small bar mixer or the rather more elaborate device known as a square-glass-jar machine shown in Fig. 11 is extremely useful. The latter device consists essentially of a stirring mechanism and a removable square-glass-jar, mounted on a frame with a motor and rheostat and stop switch. The stirring mechanism consists of a vertical shaft (*c*) carrying at the lower end a four-armed paddle (*e*). A grooved cone pulley (*a*) with hub extended to run in the brass-bushed bearing (*b*) carries on its upper face a slot which is engaged by the lug (*f*) on the lower face of the compression collar (*d*). This arrangement permits the distance to which the stirrer projects into the jar to be easily varied at will and allows the stirrer to be lifted above the top of the jar when the latter is to be removed or set in place. A variable speed $\frac{1}{4}$-h.p. motor, giving a speed range on the agitator shaft up to 2500 r.p.m. should be supplied. An ore charge of 300 to 350 gms. can be treated in this machine.

For preliminary qualitative tests by the pneumatic process, the apparatus shown in Fig. 12 is useful. It consists of a three-legged brass casting (*a*) and a brass flange (*b*) which takes a 4-in. gage glass. The flange is fastened to the gage glass by means of plaster-of-Paris. A piece of canvas or twill to serve as a porous bottom is clamped between the parts (*a*) and (*b*) by means of screw clamps. An overflow may be simply provided by cutting a hole of the proper dimensions in the bottom of a shallow pudding tin 6 ins. internal diameter and fitting the same with a discharge spout. Air is provided as for the laboratory Callow cells. The ore charge for this machine is 200 to 300 gms.

Cascade laboratory flotation apparatus is not standardized. A satisfactory apparatus consists of a cylindrical bottle of about two liters capacity with a gradual slope from full cross-section to neck, with the bottom cut off and the cut edge ground to a plane. This is set up in an inverted position, with a one-hole rubber stopper and glass tubes of various bores for regulating

the discharge, about 12 to 18 ins. below a pulp reservoir or pressure box from which pulp discharges through a tube just large enough to allow free discharge. Settling of sand in the reservoir

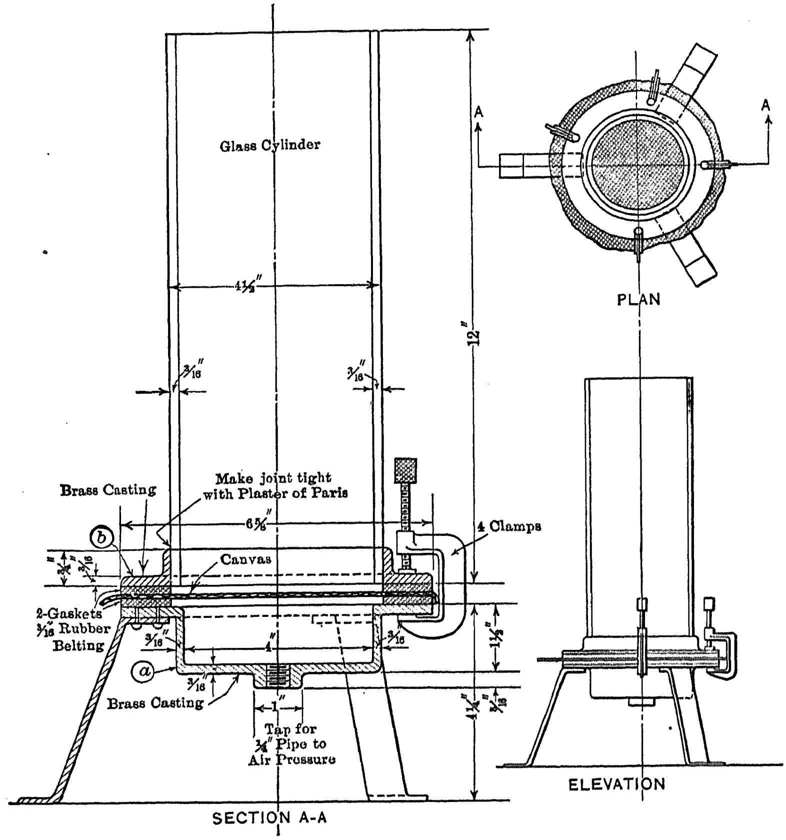


FIG. 12
Cylindrical pneumatic cell

should be prevented by slow stirring. The stream of pulp from the reservoir spigot plunging into the body of pulp in the inverted bottle carries in air which causes concentration by bubble-column action. If the inverted bottle is fitted with an overflow launder such as is described in the preceding paragraph, and if the tailing is caught in a bucket and returned manually to the

reservoir at such a rate as to make the discharge continuous, a fair quantitative result can be obtained

Motors. The five motors specified below are recommended by the Stimpson Equipment Co. for use with Janney laboratory machines, and are, of course, suitable also for use with the other types of machine where $\frac{1}{4}$-h.p. motors are specified. The writer has used two of these types of motors and has been assured by users of the other types of their entire satisfaction with them.

Alternating Current

General Electric Co. Type SCS–521–4; $\frac{1}{4}$-h.p.; 1800 r.p.m. Variable speed, reversible running. Single phase. 60 cycles. Form G–BR. Amperes 4.2 to 2.1. Volts 110 : 220. Speed, full load, 1800 to 900 r.p.m. A dial controller brake to vary the speed is supplied at an extra charge.

Kimble Electric Co., 634–646 N. Western Ave., Chicago. Single, two or three phase circuit. Variable speed, reversible running. 110 or 220 volts. Speed 500 to 2000 r.p.m. A contained lever starts or stops motor; reverses or changes the speed.

Direct Current

General Electric Co. Type SD. Constant speed. Shunt wound. Volts 110. Speed 1700 r.p.m. Field rheostat for armature circuit. Ohms 50. Amperes 3 to $1\frac{1}{2}$.

Type DSD. 1043A. Constant speed. Shunt wound. Max. volts 250. Speed 1700 r.p.m. Field rheostat for armature circuit. D.L–T. Type F. Cr. 8000. Ohms 240. Amperes 1.25 to 0.63.

Westinghouse Electric and Mfg. Co. Type CD. Shunt wound. Continuous duty. Volts 230. Amperes 1.2. Speed 1725 r.p.m. A field rheostat must be used in connection with motor.

The Robbins and Myers Co., Springfield, Ohio. List D142. Volts 110. Speed 1750 r.p.m. A field rheostat must be used in connection with motor.

In all cases specify 3-in. grooved pulley for $\frac{3}{8}$-in. round belt.

Flotation table. A convenient method of mounting flotation test machines is shown in Fig. 13. It consists of a solidly built

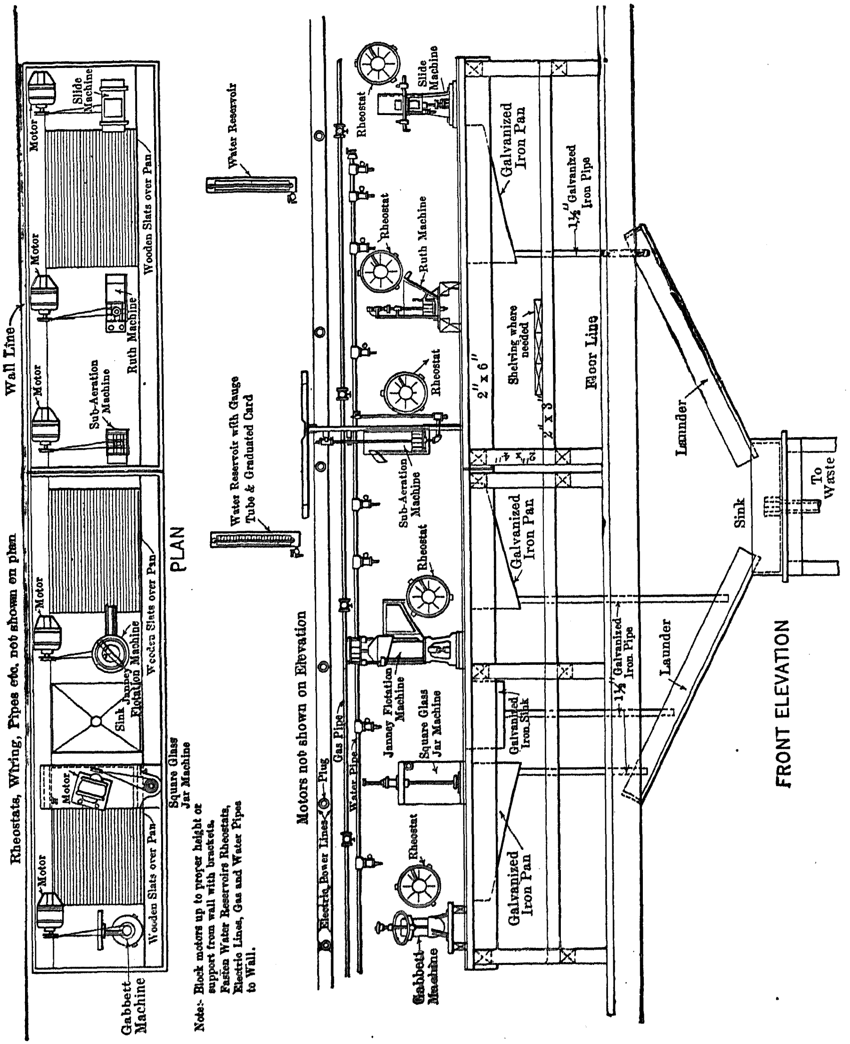


FIG. 13
Table for laboratory flotation machines

table, with 2-in. cypress top except where grids and sink are shown. The grids are made of $\frac{1}{2}$-in. by $\frac{3}{4}$-in. cypress, spaced $\frac{1}{4}$ in. in the clear. Grids and solid top are so disposed as to allow space for a machine on both sides of each grid. About

2-ft. length of solid top should be allowed for each machine. The table is best placed against a wall along which are carried power, gas and water lines with numerous convenient outlets. Where several machines are not in use at one time the number of motors may be reduced by placing a stand at each machine of proper height for the motor and placing a rheostat and stop switch on a board which bolts with wing nuts to the wall and which carries an electric plug to connect with the nearest convenient power outlet. With a device such as this a motor with its rheostat can be readily moved within 5 or 10 minutes. The water reservoirs are small galvanized tanks 3 ins. square of about 3000 cc. capacity and are fitted with gage glasses and a calibrated card reading to 100 cc. A piece of rubber tubing is attached to the faucet and provided with a pinch cock at the lower end. This makes it easy to regulate and measure the amount of water used during the test.

Flotation Agents

Principal flotation agents or "oils." The following is a list of the principal oils used in ordinary practice. Steam-distilled pine oil, turpentine, pine-tar oil, coal tar, coal-tar oil, coal-tar creosote, wood tar, wood-tar oil, wood creosote, petroleum residuum, crude kerosene, sludge-acid kerosene, *alpha* naphthylamine, xylidin, crude cresol, crude carbolic acid. Dealers are glad to supply samples of oils and the laboratory stock soon becomes large. In general it is unwise to do any extended testing work with a sample of which an insufficient amount is on hand for physical tests, because without such tests it is impossible to specify the oil accurately, if it should be desired, as a result of the tests, to buy. Proper physical tests require a minimum of ½ gallon of oil. The following is a partial list of firms which manufacture and distribute flotation oils: American Creosoting Co., Chalmette, La., (1); American Tar Products Co., 208 So. LaSalle St., Chicago, Ill., (1); Associated Oil Co., Los Angeles, Cal., (4); Barrett Co., 17 Battery Pl., New York, and in principal cities, (1); Cleveland Cliffs Iron Co., Cleveland, Ohio, (2); Denver Gas and Electric Co., Denver, Col., and gas

companies in other principal cities, (1); Georgia Pine Turpentine Co., 156 Perry St., New York, (2); General Naval Stores Co., 175 Front St., New York, (1), (2), (3); Geo. P. Jones and Co., 205 No. Lever St., St. Louis, Mo., (4); F. J. Lewis Mfg. Co., 2500 So. Robey St., Chicago, Ill., (1); National Aniline and Chemical Co., New York, branches in principal cities, (1); Pensacola Tar and Turpentine Co., Gull Point, Fla., (2); Semet-Solvay Co., Chalmette, La., (1); Standard Oil Co., branches in principal cities, (4); Texas Oil Co., Port Arthur, Tex., (4); Union Oil Co., Mills Bldg., San Francisco, Cal., (4); Utah Oil Refining Co., Newhouse Bldg., Salt Lake City, Utah, (4); Yaryan Naval Stores Co., Brunswick, Ga., (3). (The figures (1), (2), (3), and (4) indicate coal-tar products, destructively-distilled wood products, steam-distilled pine oil, and petroleum products respectively.)

Minor agents. Sulphuric acid, lime, sodium carbonate, copper sulphate, sodium sulphate, sodium hydroxide, sodium silicate, sodium sulphide, calcium sulphide. The first four are most used

Oil Testing

For testing oils the following apparatus is necessary. Specific gravity bottles 5 cc., 10 cc., 25 cc. and 50 cc. Engler viscosimeter for viscous oils and Ostwald viscosimeter (Figs. 20, 21) for mobile oils. Refractometer. Hydrometers with range from 0.7 to 1.3. Oil distillation apparatus (Fig. 23) consisting of 1000-cc. and 250-cc. distillation flasks with side tube at the middle of the neck, 360-degree thermometer accurately calibrated; copper trough condenser. One 120-cc. separatory funnel with stem graduated to 20 cc. by $\frac{1}{10}$-cc. Hand centrifuge with four Babcock cream-testing bottles, with neck graduated for 2 cc. by $\frac{1}{25}$-cc. Special tar-acid separatory funnel 100-cc. capacity, as shown in Fig. 22. Ostwald gas regulator for constant-temperature baths.

Miscellaneous Apparatus

The following items need no further description. Pulp balances, 200 gms. capacity. Pipettes (Mohr's), Fig. 14, two each, 1-cc. by $\frac{1}{100}$-cc., 2-cc. by $\frac{1}{50}$-cc., 5-cc. by $\frac{1}{10}$-cc., and 10-cc. by $\frac{1}{10}$-cc. Graduated cylinders capacity 5-cc., 10-cc., 25-cc., 100-cc., 500-cc., 1000-cc. and 2000-cc. Sugar thermometer with enclosed paper scale reading from 0° to 50° C. Speed indicator. Two doz. granite pans 8 in. by 15 in. by 2 in. Assorted beakers. Means for heating pulps and drying samples.

FIG. 14
Mohr pipette

An assay laboratory in connection with the testing laboratory is essential for any extensive work.

CHAPTER III

TESTING

Introduction. Practice in testing ores for flotation varies, of course, in its detail, from practice in testing work for other purposes, but in this as in other such work certain general principles apply. These principles would seem to be so obvious that they should not need to be enunciated, but the writer has found in going over the results of extensive series of tests done by others, and even in directing work in his own laboratory, that the rules, simple as they are, are likely to be more honored in the breach than in the observance.

The general rules follow:

1. Keep a careful, detailed record, *during the course of every test*, of every feature connected with the test, making special note of all conditions surrounding any unusual performance.

2. In keeping the records, express weights, volumes, distances and time in the accepted units, and in describing appearances and performances use a terminology that will convey a picture to a person who has not seen the work, is not familiar with the operator, and who may not be familiar, even, with the subject under investigation.

3. Keep the original record of the test neatly, in ink, or better with a medium hard pencil. Take all notes other than those recording routine tests, in a stiff-backed, tight-leaf book, of convenient size for the pocket of the operator. Date every page. When a report is made up from the original notes indicate the fact on each page so written up by an appropriate inconspicuous symbol. Differentiate in the book of original record between data entered at the time of the test and data computed or subsequently entered (such, for instance, as assays and recoveries) by making the later entries in a different colored ink or by some similar device. Have the record of each day's work or each sepa-

rate test signed by the person in charge of the test with a note as to the personnel of the testing crew and the duties of each member. Keep the original records of routine tests on printed or mimeographed forms. These forms should be the product of the best thought of the man directing the work and should be frequently changed at first until it becomes apparent that they are the best possible for the purpose of conveying to others than the operator the observed facts.

4. When more than one condition in a series of tests is capable of variation in such a way as to affect the result or affect the effect of another variable on the result, never change more than one such condition at a time in successive tests. It cannot be too strongly urged upon the laboratory experimenter that a given flotation test be carried through under the conditions prevailing at the beginning and that changes in condition during the course of the test, such as variation in temperature or speed of agitation or new additions of flotation agents, should be avoided. A test which starts in the cold with agitation of a given degree and with a certain amount of flotation agents present, and which, in the course of the test is subjected to the addition of heat and an increase in the degree of agitation and the addition of more flotation agents of the same or different variety, will, even though it may give a good recovery, give no information that is of use in the subsequent investigation. On the other hand, if the first test is carried through to a conclusion and marked as a failure, and a subsequent test is carried out under the conditions finally attained in the first test, such conditions, however, prevailing from the start of the test, then the results can be translated, without error, into mill practice. The question is the old one of making haste slowly, and the procedure advocated is by far the swiftest, as the writer can vouch after an experience with several thousand laboratory tests.

5. Never be satisfied with a doubtful result or a questionable record. If possible repeat the work, otherwise give due weight to the doubtful character of the result or record in drawing conclusions.

6. Insist upon a report by the operator immediately following

the completion of each series of tests, or, in an extended series, at not longer than weekly intervals. Work up the full results of a series as soon as it is completed while the matter is fresh in mind.

The variables affecting froth flotation were discussed in Chapter I. They are enumerated in condensed form below:

1. Ore.
    - (*a*) Mineralogical character.
    - (*b*) Fineness of grinding.
    - (*c*) Method of grinding.
2. Agents.
    - (*a*) Principal flotation agent. ("Oil")
        - α Character.
        - β Quantity.
    - (*b*) Minor agent.
        - α Character.
        - β Quantity.
3. Water.
    - (*a*) Quantity with respect to solids, i.e. pulp thickness.
    - (*b*) Character.
        - α Acidity or Alkalinity.
        - β Flotation agents.
        - γ Dissolved salts.
4. Apparatus
    - (*a*) Method of aëration.
    - (*b*) Method of froth removal.
5. Agitation.
6. Duration of treatment.
7. Temperature.

The variable results to be watched for and recorded are:

1. Recovery.
2. Grade of concentrate.
3. Rate of flotation.

4. Froth.
   (*a*) Copiousness.
   (*b*) Consistency.
   (*c*) Size of bubbles.
      α At pulp surface.
      β At froth surface.
   (*d*) Solid load.
5. Appearance of tailing.
   (*a*) Flocculation.
   (*b*) Settling rate.

A useful form for recording tests follows.

RECORD OF FLOTATION TEST

Series No..........

Test No...........

FEED

Origin...

Method of treatment from time of cutting sample until used for test.............

....

....

Approximate mineral composition...

....

Total weight of charge, solids,..gm.

Assay Analysis:

Number........................ % % % % Insol.

Sizing Test

No.........

Size	Per cent. weight material	Per cent. weight material cumul.	Assay per cent. metal	Micros. per cent. sulf.	Pounds per ton of feed		Pounds sulfide per ton of feed	
					Direct	Cumul.	Direct	Cumul.
TOTALS								

Reagents

Name	Laboratory Number	Per cent.	Sp. gr.	CC.	Lbs. per ton

State order of adding reagents...
....

Water

Source..
Litmus reaction...

Machine

Type used...

RECORD OF MANIPULATION

Pre-agitation period:

Duration..min.
Speed..r.p.m.
Quantity of water added..cc.
Per cent. solids...

Roughing — Concentrate period (Underline which one)

Duration..min.
Speed..r.p.m.
Quantity of water added..cc.
Weight of concentrate, wet..gm.
Weight of tailing, wet...gm.
Weight of untreated material (wet)............gm. (Added to cleaner tailing)

Froth (Underline proper descriptive term)

Texture: Even, uneven, coarse, fine

Bubbles:

	Beginning	Middle	End
Size at water line, inches.........			
Size at lip line, inches...........			
Character..........................	Elastic Effervescent	Elastic Effervescent	Elastic Effervescent

Quality: Viscous, tender
Mineralization: Heavy, medium, light; gangue, sulphide
Aëration: Thickness of froth in machine:
At beginning of period..in.
At middle of period..in.
At end of period..in.
(Measure thickness from lip line to water line)
Persistence: High, medium, low
Microscopic examination:..
Litmus reaction:..
Remarks..
....

Pulp

Litmus reaction..
Degree of flocculation..
Rate of settling in spitzkasten..............................in. per min.
Per cent. solids..

Cleaner — Middling period (Underline which one)

Duration..min.
Speed..r.p.m.
Quantity of water added..cc.
Weight of concentrate, wet..gm.
Weight of tailing, wet..gm.

Froth (Underline proper descriptive term)
Texture: Even, uneven, coarse, fine
Bubbles:

	Beginning	Middle	End
Size at water line, inches.........			
Size at lip line, inches...........			
Character.....................	Elastic Effervescent	Elastic Effervescent	Elastic Effervescent

Quality: Viscous, tender
Mineralization: Heavy, medium, light; gangue, sulphide
Aëration: Thickness of froth in machine:
At beginning of period..in.
At middle of period..in.
At end of period..in.
(Measure thickness from lip to water line)
Persistence: High, medium, low
Microscopic examination:..
....
Litmus reaction:..
Remarks..
....

Pulp

Litmus reaction..

Degree of flocculation..

Rate of settling in spitzkasten................................in. per min.

Per cent. solids..

TEMPERATURES

At beginning ....Degrees C.

At end of pre-agitation period.............................. "

At end of rougher period .... "

At end of cleaner period .... "

METALLURGICAL RESULTS

	Weight of product, gm.	Per cent.	Per cent.	Per cent.	Per cent. insol.
Heading..................					
Tailing....................					
Concentrate...............					
Middling..................					
Untreated material.........					

Ratio of concentration............. ....

Per cent. indicated extraction........ ....

Remarks...

....

....

Date..................... Figured by......................

Operator.................. Checked by......................

Approved by...................

NOTES ON DESCRIPTIVE TERMS

Texture:

Under this head the grain of the froth should be examined. Mental reference to the grain of a rock or other non-homogeneous mixture will help in choosing the proper descriptive term.

The texture will be described as "even" when all or a great majority of the bubbles on a given horizontal plane are of the same size. "Uneven" describes the reverse of this condition.

The texture will be described as "fine" when all or a great majority of the bubbles at $\frac{1}{2}''$ above the water line are $\frac{1}{8}''$ or less in diameter. "Coarse" will be used to describe textures coarser than the above.

Character of Bubbles:

Under this head the behavior of the bubble films themselves should be noted.

The character is "effervescent" when the bubbles burst with considerable violence soon after reaching the surface.

The character is "elastic" when the bubble films are rather persistent on the spitzkasten, when they may be deformed considerably without causing them to burst and when they tend to elongate markedly in overflowing the lip of the spitzkasten.

Quality:

Under this heading the nature or character of the froth as regards its behavior as a mass should be described.

The quality will be "viscous" when patches of the froth act almost as solids and the whole froth is sluggish on the cell.

The quality will be "tender" when the reverse of this condition holds, i.e.: the froth is homogeneous, fluffy, motion in one part is not transmitted to any distance and the froth as a mass is active on the surface of the cell.

Mineralization:

Under this head should be described the solid load in the bubble film. The mineralization is "heavy" when this solid load is great. It is light when the reverse is true. In general a heavily mineralized bubble will be practically opaque; a lightly mineralized bubble will be transparent to slightly translucent. "Medium" mineralization will describe the intermediate condition.

The word "gangue" or "sulphide" should be underlined in this connection when either predominates in the solid load in the bubble film.

Persistence:

"High" persistence is indicated when the froth in the collection pan shows slight or no tendency to break down after ten minutes standing. "Medium" persistence is indicated when the volume of froth spontaneously breaks down to somewhere in the neighborhood of half its volume within ten minutes. "Low" persistence is indicated by practically complete breaking down on ten minutes standing.

This form is conveniently made up in the form of a sheet 17 ins. by 11 ins. folded to 8½ by 11 and punched for a loose-leaf holder on the folded margin. With such an arrangement page one runs to the heading "water," page two to the heading "Cleaner-middling Period," page three completes the record form, and the "Notes on Descriptive Terms" are printed on the back or fourth page.

Testing axioms. 1. Any ore amenable to froth flotation can be treated with equally good metallurgical results by the agitation-froth or the bubble-column process. Change in process will, however, usually require a corresponding change in some other essential condition of the flotation treatment.

2. Any result that can be obtained in a laboratory machine can be duplicated on a mill scale. Whether or not the duplication of the controlling laboratory conditions in the mill can be profitably made is another matter. In general, however, such duplication is economically possible; indeed mill conditions of treatment are generally more favorable to good flotation results than laboratory conditions.

Procedure in Flotation Tests

General. Always thoroughly clean the flotation machine and the apparatus used for measuring reagents before starting any test which is intended to give information concerning the behavior of a certain flotation agent or mixture of agents. The machine is best cleaned as follows: Scrub with a strong solution of sodium carbonate, fill the machine with water and run for a short interval, draw off the water, refill the machine with clean water, add about 5 cc. of concentrated sulphuric acid, again run for a short time, draw off the water, add fresh water and a charge of waste rock ground to flotation size and again run. If no frothing occurs the machine is sufficiently clean. For routine tests where the same flotation agent is being used in successive tests it is only necessary to run the machine with a charge of the cleaning solid and water and observe lack of frothing on agitation. Of course, if any considerable amount of frothing occurs on such agitation, the apparatus should be cleaned as above before proceeding. Pipettes, burettes, etc., should always be thoroughly cleaned with oil solvents and chromic acid solution before being used.

Measurement of quantities for tests. Quantities of reagents in flotation practice are stated in this country in pounds avoirdupois per short ton of ore. It will be found simplest in test work to use metric units for measuring solids, water and reagents. The charge of solids for a test will then be weighed in grams and the water and reagents will, in general, be measured in cubic centimeters. Mobile oils and acid are best measured with a Mohr pipette (see Fig. 14). Viscous oils are measured by counting the number of drops added to the machine, a determination

having been made of the number of drops per cc. and the specific gravity, or directly of the number of drops per gram. Solid reagents are, of course, weighed in, if added as solids, or the number of grams per cc. determined and the number of cc. measured if the reagent is added as a solution. Considerable time and calculation can be saved by making up a solution of sulphuric acid which shall contain 1 gm. of sulphuric acid per cc. The specific gravity of oils ranges, in general, between 0.9 and 1.1. For determining the number of cc. to be added, the calculation involving the specific gravity must, of course, be made if it is desired to add an accurate quantity. For ordinary testing work, however, it is sufficient, in calculating in the laboratory the amount of oil to be added, to take the specific gravity as one, and later make the accurate calculation of the actual amount used in pounds per ton.

The chart, Fig. 15, will save much time in calculation of reagent quantities. The method of use is as follows: Example (1): Knowing the weight of ore charge, say 500 gms., and the density of the oil or other agent, say 1.0, to find the number of cc. of oil or agent necessary to add in order to have present some given number of pounds per ton of ore, say 20; join, with a straight-edge, the points 500 on the scale reading "Weight of Ore Charge in Grams" and 20 on the scale reading "Pounds of Oil per Ton of Ore" and mark the intersection with the line "X–X"; now join this point of intersection with the point 1.0 on the scale "Density of Oil" and at the intersection of the prolongation of this line with the scale reading "Cubic Centimeters of Oil" read 5.0, which is the volume of reagent that must be added to fulfill the required condition. Example (2): Given weight of charge as 500 gms., density of agent as 1.0, and, having added 5.0 cc., to find the number of pounds of agent per ton of solids that this represents; join the point 1.0 on the density scale with the point 5.0 on the volume scale and mark the intersection on "X–X"; then join this point of intersection with the point 500 on the weight-of-charge scale and at the intersection of the prolongation of this line with the pounds-of-oil scale read 20, which is the result desired.

Agitation froth machines without froth overflow. Machines of this type are the square-glass-jar, the Gabbett mixer and similar apparatus. In these machines the purpose in a test is to agitate the pulp with the flotation agent in such a way as to cause precipitation of air bubbles on all of the sulphide mineral particles and subsequent coalescence of the bubbles with their attached loads to form the heavily coated bubbles and bubble agglomerates characteristic of the agitation-froth process. The procedure is as follows: Place in the machine an amount of water sufficient, with the solid charge, to make a pulp containing 20 to 25 per cent. solids. Start the impeller slowly and add the solid charge. Add flotation agents in proper amounts, taking care that the agents get into the pulp and do not stick on the sides of the containing vessel or on the agitator shaft. Raise the speed of the impeller to that desired (1000 to 2500 r.p.m. is the usual range) and continue agitation for a period of, in general, from 6 to 10 minutes. The determination of the proper end point is in some cases extremely important, as, if the agitation is stopped too soon or carried forward too long, mediocre results only will be obtained. The proper end point can be determined by taking small samples of pulp from the machine by means of a pipette and examining them on a watch glass under a hand glass or microscope. It will be noticed in such examination that at first little or no change in the appearance of the pulp is observable. Shortly, however, small bubbles, each carrying but one or two easily visible pieces of sulphide mineral, are to be seen. Subsequent samples show an increase in the number of solid-loaded bubbles and an increase in the load on the bubbles. Finally agglomerates of solid-coated bubbles appear. At this point agitation should be stopped in order to get the maximum amount of froth. In the case of some flotation agents, such as oleic acid in a pulp containing sulphuric acid, the progress of the experiment may be watched by the color of the pulp, which changes, as the sulphide particles become agglomerated with air bubbles, to that of the gangue minerals with specks the color of the sulphide mineral, indicating the coated air bubbles. The froth which rises on cessation of agitation may

be removed by introducing water into the bottom of the machine in order to raise the pulp level until the froth overflows, or by skimming with a spoon. The former method is the better. In case no provision is made in the machine for introduction of water at the bottom, this may be accomplished by means of a long-spouted funnel introduced from the pulp surface. A full record should be kept of all phenomena observed, using the form on page 53 or a similar outline as a guide.

Slide machine. This machine was devised to allow of intermittent agitation and successive removals of froth. The procedure in a test is as follows: Introduce into the machine an amount of water equal in weight to that of the solid charge. Start the impeller slowly and introduce the solid charge. Add flotation agents in proper amounts. Raise the speed of the impeller to that to be employed in the subsequent flotation operation (1200 to 1800 r.p.m.) and agitate for about one minute to obtain thorough mixing of the flotation agents with the pulp. Add water to bring the pulp level up to the top of the lower compartment. Start the impeller and continue agitation for about two minutes. Stop the impeller and allow froth to rise. Remove the clamps and slide the froth obtained into a pan. Replace the top, again add water to raise the pulp level to the top of the lower compartment, agitate for another period of two minutes and again remove froth. Repeat this operation as often as is necessary to obtain a clean tailing or until further repetitions produce no apparent improvement in the final result. The various froths removed will correspond very roughly to those to be expected from successive cells of a mill-sized Minerals Separation machine working on the same ore and under the same conditions, the two-minute intervals allowed corresponding to the average time of passage of pulp through a mill-sized flotation cell. Note, however, that a mill operation will not be subject to the same variations in percentage of solids in successive cells that are present in this test. Use form on page 53 to record observations.

Janney laboratory machine. Place in the machine an amount of water equal in weight to that of the solid charge to be used.

Start the impeller slowly. Add the solid charge. Add the flotation agents. Place the cover on the machine in an inverted position, bring the impeller up to speed (1800 to 2500 r.p.m.) and agitate for one minute. Remove the cover and wash the adhering pulp down into the machine. Place cover on machine in an erect position and add water until the pulp level in the froth-separating compartment is within one inch of the level of the overflow lip or, in the case of a test in which there is considerable frothing, until the froth begins to overflow without help. Carry forward froth formation and removal for a length of time to be determined by the appearance of the froth removed and the residual pulp, bearing in mind that each two-minute interval corresponds roughly to the passage of the pulp through one cell of a mill-sized Janney mechanical machine. During the test keep pulp level up to necessary height by addition of water. In case it is desired to test for a mill installation which is to make finished concentrate on the early cells and middling on the later cells, the froth removed at successive two-minute intervals should be caught separately and later combined according to the judgment of the operator, or assayed separately. In case the test is to give information concerning the performance of a pulp in a flow sheet of the rougher-cleaner variety (see page 139) in which a rough concentrate is to be taken from an early machine and cleaned in a later machine, all of the froth collected in the first frothing operation is re-treated in the machine after the residual pulp from the first operation has been removed. In such a case the residual pulp from the first frothing operation represents the rougher tailing to be expected, the finished froth of the second operation represents the cleaner concentrate, and the tailing from the second operation represents the cleaner tailing or middling of the process which would be returned to the circuit. The record form on page 53 is designed particularly for tests in this type of machine.

Minerals Separation machine. Procedure in this machine is similar to that described above for the Janney machine except insofar as concerns the manipulation of the cover, which latter is lacking in the case of the Minerals Separation machine.

Pneumatic machine. In ordinary mill operation of pneumatic machines the flotation agent has usually been pre-mixed with the feed to the machine. One method of pre-mixing is to add the flotation agent to the grinding mills. If such procedure is employed or contemplated in the mill, then for laboratory tests, a charge of ore, dry-ground somewhat coarser than is desired for the flotation operation, should be mixed with the proper amount of flotation agents and enough water to make the consistency 1 : 1, and ground in a ball or pebble mill. If sulphuric acid is to be used, a silex-lined pebble mill should be employed. The duration of the grinding should be sufficient to bring the pulp to a size suitable for the flotation operation. The charge of thick pulp should next be thinned by the addition of water and then be introduced into the flotation machine with the air turned on and the tailing exit closed. From this point the procedure varies according to the character of the test installation. If a non-circulating test is to be run, the tailing exit is kept closed for the duration of the test and sufficient air is turned on to cause overflow of froth without any considerable disturbance of the pulp. The air should also be regulated in the different compartments so that there is no marked eddying from a center over any given compartment. The frothing period should be continued until the tailing shows a marked impoverishment or until 8 to 10 minutes have elapsed. This time corresponds to that required for the passage of pulp through an ordinary Callow machine under normal operating conditions. If, at the end of this time, no marked impoverishment in the tailing is noticeable, it may be concluded that the particular combination of ore and flotation agent is unsatisfactory. In case the apparatus shown in Fig. 9, with pulp circulation, is to be employed, the procedure recommended by the General Engineering Co. is as follows: Open valves (*A*), (*B*), (*B′*) and (*D*) and crack valves (*C*) and (*C′*). Put about two inches of clear water in the cleaner cell and wet down the sides and blanket of the rougher cell. Pour the prepared sample into the rougher cell and open valve (*C*) until froth starts to overflow. If the froth overflow is uneven, adjust the valves (*B*) until a satisfactory

overflow is obtained. Thereafter regulate the air supply entirely by valve (*C*). Adjust the air supply in the cleaner cell in like manner. Turn cock (*F*) to allow pulp to discharge into the air lift. Carry the water level in the rougher cell at from three to five inches below the overflow lip, and in the cleaner cell at from two to four inches below, adjusting the level in the rougher cell by the addition of water and in the cleaner cell by regulation of the siphon (*E*). Continue circulation of the pulp as long as mineral-bearing froth is discharged from the rougher cell. This should not be more than 10 to 20 minutes in case of a proper combination of flotation agents. A few minutes before the end of the test the material in the air lift should be collected by pulling the cork (*I*) and draining into a bucket. This material should be poured directly into the rougher cell. At the end of the test remove the tailing from the rougher cell by turning the cock (*F*) so that it discharges through the pipe (*H*) into a bucket. Wash the blanket thoroughly. Collect pulp from air lift. Collect cleaner tailing by manipulating the siphon (*E*). The blanket in the cleaner cell should be cleaned by washing with water. The tailing from the rougher cell will represent the rougher tailing from a mill plant run under the conditions of the test, the cleaner concentrate the corresponding product from the mill, and the cleaner tailing the circulating middling. The General Engineering Company presents the following method of physically disposing of the cleaner tailing: "Plaque out the mineral and add it to the original concentrates already obtained, the remainder being added to the rougher tails; . . . ."

Procedure in other laboratory pneumatic machines may easily be inferred from the detailed instructions just given.

Vacuum machine. For a test in this machine a charge of 300 to 500 gms. of ore is added to an equal weight or less of water, the quantity of water being about sufficient to make a pulp of the consistency of fairly thick cream, and to this is added oil in an amount equal to from 1 to 5 lbs. per ton. The oil used may be any of the oils common to present-day flotation practice or a fatty or mineral oil. The choice of oil will depend to some extent on the kind of ore to be treated. The mixture of

pulp and oil is thoroughly stirred to insure dispersion of the oil throughout the pulp. Stirring for several minutes with a rod or spatula, or for a shorter length of time in an apparatus such as the square-glass-jar machine, is sufficient to accomplish the required dispersion. In some cases it may be advisable to add sulphuric acid, and in some cases to heat the pulp to between 30° and 40° C. The apparatus is now filled with water to a depth of about 2 inches, the valve (*A*) is closed, and a suction of about 5 inches of mercury is applied. The pulp is next poured into the funnel (*B*) and the delivery into the separating chamber is regulated, by means of the valve (*A*), in such a way as to prevent inrush of air. After all the pulp is in the separating chamber, water is added until the level of liquid in the inverted funnel stands at 1 to 2 inches below the neck. The vacuum is now increased gradually and the rake started revolving at from 10 to 12 r.p.m. Agglomerates will be seen to form, most easily in the sand at the bottom of the apparatus, and to rise to the surface of the liquid, forming there a heavily mineralized, stiff froth. This will gradually build up until it overflows into the dome (*J*) and through the pipe (*E*) into the concentrate collecting vessel (*F*). In the course of 20 to 30 minutes the vacuum is increased to 22 to 26 inches of mercury. Water is added at intervals to replace the water overflowed in the form of froth. At the expiration of the flotation period the vacuum is released, the valve on the tailing discharge pipe is opened, the tailing is drawn off and the dome (*J*) is removed and washed free from settled concentrate.

Potter-Delprat process. No special apparatus is needed for a test of this process. A funnel with preferably about 30 degrees apex angle and 300 or 400 cc. capacity is sufficient. This is connected by means of rubber tubing with an acidproof reservoir set so as to have a head of 4 or 5 ft. on the point of discharge into the funnel. The charge of moist ore is placed in the funnel and hot acid or acid salt cake solution (in the case of sulphuric acid, with a strength of from 1 to 10 per cent.; in the case of acid salt cake solution, having a density of 1.3 to 1.4) is slowly added. Almost immediately upon the addition of acid, agglo-

merates of air bubbles and sulphide mineral particles will be seen to form in the body of the pulp and gradually these will rise to the surface to form a thick, coherent, heavily mineralized froth. If the capacity of the funnel is sufficient to allow for several minutes collection of froth before overflow takes place, a froth from $\frac{1}{2}$ inch to 1 inch thick will easily build up. After the liquid level has reached the overflow rim of the funnel and liquid begins to overflow, there is, of course, no opportunity for froth to build up, and consequently the mineral will come over as agglomerates not collected together in the usual froth form. A somewhat more elaborate apparatus may be made by reproducing on a small scale the pointed box shown in Fig. 27. Such a box, about 10 or 12 inches deep with one or two sides made of glass, with a top-regulated ball valve for tailing discharge, can, with care, be operated to give a good idea of mill practice with this process.

Continuous tests with machines of the size described in this chapter will not give satisfactory indications of what can be expected from mill operations.

The subject matter of most flotation tests will fall under one of the following heads:

1. Amenability of an ore to flotation.
2. Process of treatment for an ore.
3. Flotation agents.

Amenability of an ore to flotation is best tested for by a microscopic examination of the ore followed by a few laboratory flotation tests. As stated in Chapter I, most ores containing minerals of metallic, resinous or adamantine luster associated with minerals of earthy, vitreous, or pearly luster, can be divided by froth flotation in such a way that the floating part will contain the bulk of the mineral or minerals of metallic, resinous or adamantine luster to the practical exclusion of the associated minerals, while the part not floated will consist largely of the associated gangue minerals. For microscopic examination, thin sections of the ore, such as are prepared for petrographic work, or polished sections should be examined through a suitable

microscope to determine the method of occurrence and grain size of the valuable mineral. Fragments of the ore crushed to pass a 0.295-mm. screen (or its equivalent) should be examined under a binocular microscope at 20 to 50 diameters magnification and also under the petrographic microscope. The examination of the fragments will give a good idea of the mineralogical character of the ore. Rogers' "Study of Minerals" (see page 23) is a good guide for this determination. The examination of thin or polished sections will tell whether alteration has taken place at the surface of the minerals that it is desired to float. If microscopic examination shows unaltered sulphides associated with the ordinary rock-forming gangue minerals, it is safe to conclude that the ore is amenable to flotation.

This conclusion can probably be confirmed by a few laboratory flotation tests as follows: Machine — Janney laboratory flotation machine. Ore — 500 gm. ground to pass a 0.295-mm. screen. Flotation agents — About 0.3 cc. of oil.* Follow the procedure described on page 61. The appearance of the froth through the glass sides of the separating compartment compared with that of the pulp will indicate whether concentration is going forward. A better measure is afforded by a microscopic examination of the froth or by examination of the same by vanning on a plaque or watch glass.

If the concentrate obtained is low-grade, it can probably be improved by increasing the percentage of moisture (lessening the weight of solids charged) or by adding petroleum residuum or low-grade kerosene (stove oil) to the oil mixture, or, except in the case of a carbonate gangue, by the addition of sulphuric acid in the proportion of from 4 to 10 lb. per ton of solids. If recovery is low due to dropping of sulphides from the froth, which will be apparent from examination through the glass side of the separating box during the progress of the test, it can probably be improved by decreasing the proportion of pine oil in the oil mixture or by increasing the ore charge. If the froth is too effervescent decrease in pine oil or addition of petroleum or wood-tar oils will add stability. Finer grinding may also aid,

* See notes on page 70.

or the addition of sulphuric acid. Reference to the following tabulation of oils and other principal flotation agents classified according to properties will serve as a guide to the oil changes. In any case, if the preliminary microscopic examination has revealed a mineralogical composition of the type amenable to flotation, persistence in the search for proper operating conditions should result in the discovery of a suitable method.

Table of Principal Flotation Agents Classified According to their Characteristic Tendencies in Flotation

Agents which tend to produce a voluminous froth.
Essential oils (steam-distilled pine oil, eucalyptus oil).
Tar acids (phenols, cresols).
Amyl acetate and amyl alcohol.

Agents which tend to enhance selection of sulphide minerals.
Petroleum and petroleum derivatives.
Coal tar and coal-tar oils.
Wood tar and wood-tar oils and creosotes.
Nitrogenous bases from tars.

Agents which tend to make stiff froths.
Wood tar and wood-tar oils.
Petroleum and petroleum derivatives.

The above list does not pretend to be complete. Neither is it to be understood that the classification is a rigid one. Thus while a considerable volume of froth, and a certain degree of stability of the same together with selection of the sulphide mineral are essential to all successful flotation operations, one oil, as for instance pine oil or coal tar, may serve for all three purposes. In general, however, best results will be obtained by a proper admixture of two or more agents.

Insofar as we know at the present time, and the experimental data are convincing, a flotation oil which is to be depended upon for froth formation must be soluble or possess an appreciable soluble portion, and the dissolved substances must have the power of lowering the surface tension of water when added thereto in small quantities. If the oil is to be depended upon for sulphide selection it must adsorb strongly and rapidly at the surfaces of substances of metallic, resinous and adamantine luster and feebly or not at all at the surfaces of ordinary gangue

minerals under the conditions of froth-flotation. The effective properties of the agents which stiffen froths are not so well known. It is probable, however, that they adsorb strongly at bubble surfaces and that when spread out in thin films their viscosity is high.

Tests for a process for treatment of an ore will always proceed most rapidly if started by a careful microscopic examination which will furnish the following information:

(*a*) Mineralogical composition.
(*b*) Petrographic structure.
(*c*) Degree of alteration of the surface of the mineral to be floated.
(*d*) The size to which the ore must be crushed in order to free a considerable part of the mineral to be floated from the associated gangue.

This information being at hand, systematic testing should be started with the following facts in mind:

(*a*) In the laboratory the agitation-froth process is easier to control and tests are more quickly run than by the pneumatic process.

(*b*) An ore that can be successfully concentrated by flotation in an agitation-froth machine can be successfully concentrated, with certain changes in the accessory details of operation, in a pneumatic machine.

(*c*) Before a mill is built the process worked out in the laboratory should be tried out on something approximating a mill scale in a test plant.

(*d*) The flotation process operates most easily and with greatest leeway on a pulp containing from 15 to 20 per cent. solids. On the other hand, power consumption, mill equipment and oil consumption are lessened as the percentage of solids in the pulp is increased.

(*e*) A change in the oil mixture in an operating mill may be a serious matter, involving considerable laboratory experimental work and costly interference with mill operation. Hence the oil chosen should be one of which a supply at a fair price is reasonably assured, and the oils tried in the testing work should

be of this class. The important members of the class are: pine oil, coal tar, coal-tar creosote, wood tar, wood-tar creosote, petroleum residuum, and the low-grade kerosene commonly known as stove oil. All of these substances are available at fair prices and in good supply practically everywhere. Certain other substances may be locally abundant and their use, temporarily at least, may be justifiable on that score, but a suitable flotation agent consisting of one or more of the above mentioned substances should be determined and the best available supply investigated against the time when the supply of the local substance is exhausted.

No general rule can be given as to the kind of oil applicable to a given class of ore. Certain general tendencies are, however, to be observed from a study of the kind of oil used in connection with certain ores at various mills throughout the country. In a majority of the mills treating ores containing chalcocite, coal tar or coal-tar creosote forms a considerable part of the oil mixture, with pine oil usually present in minor quantities. In the mills in which galena or sphalerite or both are recovered, wood tar or wood-tar creosote is used in a considerable number of cases. In the treatment of some such ores, pine oil and crude kerosene are employed. In mills using 20 to 25 lb. of oil per ton of ore, petroleum residuums form a large part, i.e. upwards of 80 per cent of the oil mixture, with pine oil or coal-tar creosote, or both, forming the balance of the mixture.

The practice in the addition of flotation agents other than oils is no more subject to generalization with regard to different types of ores than is the question of the kind of oil. The only general statement that seems to be founded in experience is that a small amount of copper sulphate solution, say such an amount as will introduce into the pulp somewhat under 0.1 lb. of copper per ton of solids, seems to improve results with sphalerite ores. Sulphuric acid is usually employed with blende ores. Sodium silicate is sometimes used where preferential separation of galena from sphalerite is practiced. A majority of the tonnage of low-grade chalcocite ores is probably being treated in neutral or slightly alkaline circuit, but the exceptions

to this rule are so numerous and the results obtained on such ores in acid circuits are so excellent, that no general rules can be formulated. Acid salt cake is used in place of sulphuric acid in some instances, notably where it is considerably cheaper or where there is a considerable amount of carbonate in the ore and treatment in an acid pulp seems to be necessary. It is not unlikely, however, that in the latter case, further investigation would reveal the fact that an acid pulp was unnecessary and that the use of both salt cake and sulphuric acid could be done away with.

The great majority of ores that can be concentrated by flotation will give a good flotation result with a mixture of pine oil and coal-tar creosote, the pine oil forming from 5 to 50 per cent. of the mixture. In laboratory tests in a mechanical machine from one to two pounds per ton of ore of such a mixture is necessary and sufficient.

The procedure for preliminary testing, then, should be as follows:

1. Grind the ore, best dry, so that all will pass the screen whose aperture corresponds to the average maximum size of grain of the mineral to be recovered, but in any case so that all will pass a 0.3-mm. screen. The grinding should further be such that at least 50 per cent. of the material will pass a screen whose aperture is one-quarter that of the limiting screen.

2. Use an agitation-type machine which will give a peripheral speed of impeller tip of from 1500 to 2500 f.p.m. in the case of a three- or four-inch impeller, slightly less in case of a larger machine. The machine should be one that can be easily and thoroughly cleaned. The Janney laboratory flotation machine rigged with the shaft extended through an overhead bearing (see Fig. 4) driven by a variable speed $\frac{1}{4}$-h.p. motor is, in the writer's opinion, the best agitation-type laboratory machine.

3. Use such a quantity of ore as will give about 20 per cent. solids at the beginning of the test. The solids-water ratio will, of course, grow less as the test proceeds and concentrate is removed, since the volume of pulp in the machine must be kept up by the addition of water.

4. Follow now the procedure of the tests outlined on page 61, using the pine oil-coal-tar creosote mixture, recording all data and observations on each test and carrying each test through at least to the point where assay samples of the products are on the shelves available for assay. It will be wise for the inexperienced operator to assay samples from many of his early tests although he may not believe that the results warrant the labor and expense. It is told by the engineer who directed the experimental work in the early days of flotation at one of the big mills of the country, that he and his staff watched the operation of a pneumatic cell for several days and never considered the work sufficiently good to warrant an assay, but allowed the concentrate to flow with the tailing to waste. Finally they decided that nothing could be done with the machine and he, merely for the purpose of obtaining a definite figure for report purposes, took a sample of the tailing for assay. The assay report showed the tailing to be considerably better than any that had been made on the agitation-type machines running in the mill. Further work, now with assays to indicate the character of the results, confirmed the first assay and demonstrated the utility of pneumatic treatment for the ore in question. The moral need not be pointed.

When a satisfactory flotation mixture that can be used as a standard has been discovered, as it should be in the course of ten or fifteen tests unless the ore is extremely difficult to float, further laboratory investigation should be carried out along the following lines: treatment in a bubble-column machine; determination of the maximum percentage of solids that will give a good grade of concentrate and a satisfactory recovery; the effect produced by the substitution in whole or in part of cheaper flotation agents; the minimum amount of flotation agent that can be used.

As to the first of these lines of investigation, power and flotation agent consumption on a given ore are generally less in a bubble-column machine than in an agitation-type machine and the operation of the former is both more simple and more flexible. It is the author's opinion that ten years' time will see the agita-

tion-froth method of flotation almost completely displaced, except in the smallest mills, by some bubble-column method.

For investigation in the laboratory of the method of treatment of the ore in question by the bubble-column process, the machine shown in Fig. 10 is wholly satisfactory. The charge of ore is best ground with the flotation agent in a laboratory ball mill in the presence of an equal weight of water. The solid charge in a machine of the size shown should be about 1500 gm. Sufficient water should be added to make the pulp in the cell about 20 per cent. solids. The air quantity should be so regulated as to maintain a gentle overflow of froth, adding water as the test proceeds to keep up the bulk of the pulp to such a point that the froth overflow can be maintained without excessive disturbance of the pulp in the cell. Use the flotation agent discovered to be best in the agitation cell. Continue the treatment until it becomes apparent that the recovery being made is not commensurate with the time spent in making it. Continue with tests of this character until a standard of performance for the machine has been set up.

Attempt now a substitution of petroleum for a part of the pine oil in the flotation-agent mixture. The apparent result should be decreased effervescence of the froth and a dirtier concentrate. The decreased effervescence need cause no worriment unless an excessive amount of air is necessary to maintain a froth overflow, and the dirtiness of the concentrate can probably be corrected in the inevitable cleaning operation. The object sought in these tests should be a tailing of the same grade as that obtained with the standard mixture. If the diminution of the pine oil decreases frothing too greatly it may be possible to get the frothing effect more cheaply with coal-tar creosote or wood-tar creosote. This investigation should be continued until the mixture containing the greatest possible proportion of the cheaper agents, which will produce a recovery and grade of concentrate approximating that obtained with the more expensive standard, is found.

Investigation should now be directed toward the effect of increasing the percentage of solids in the pulp. Such increase will,

as previously stated, result in an increase in the tonnage that can be handled through a given machine in a given time, thus saving installation and power expense. It will also decrease the amount of flotation agent necessary and may make possible coarser grinding. The first apparent effect of an increase in the percentage of solids in the feed will be a dirtier concentrate. This can, however, be easily cleaned up in the more dilute pulp in the "cleaner" machine.

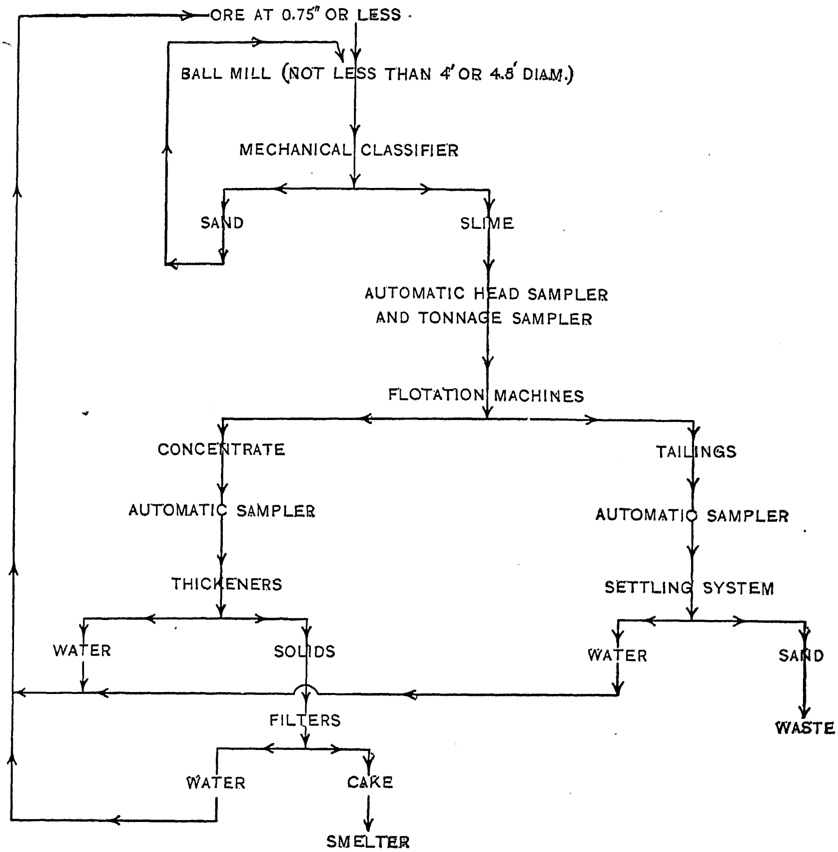


FIG. 16
General flow-sheet of test plant

When the laboratory investigation has proceeded thus far the methods developed should be tested out in a test mill, built

at the plant, using the water that is to be used in the final plant.

A satisfactory test mill should be capable of treating 50 to 100 tons per day and should have a maximum of flexibility, with provision for taking tonnage and assay samples with the least inconvenience and interference with results. While each test mill will vary from every other in its details, the general flow sheet and arrangement of the flotation part of a test mill should be approximately as shown in Fig. 16.

Oxidized ores. By oxidized ores, as the term is commonly used in flotation practice, are meant the oxides, carbonates and silicates of lead, zinc and copper. Tin oxide, cassiterite, also offers a problem, although a different one, in all probability, from the others.

Methods proposed for the treatment of oxidized ores group themselves under three different heads: (1) direct flotation, (2) those depending upon the formation of a film of a compound of metallic luster on the surface of the valuable oxides and (3) those dependent upon the solution of all of the valuable oxides and their subsequent precipitation as metal or sulphide, i.e. as compounds with a metallic luster. All of the obvious chemical methods of filming and of dissolving and re-precipitating have been tried and patented. Some of the patented methods work with given ores but none are generally successful. No method of procedure that promises successful results can, therefore, be set forth here. For those, however, who are going to embark on the search it may be said that experience points to success along the lines of solution and reprecipitation or of direct flotation, rather than of filming. Copper carbonates can be floated apparently as such from ordinary gangues in the form of a low-grade concentrate in such an amount as to make a real recovery. Also metallic copper and copper sulphide precipitated from solutions of copper salts can be floated. Precipitation is, however, hard to regulate in the presence of the associated pulp and subsequent flotation also presents considerable difficulty.

Differential and preferential flotation. The pressing problems in differential and preferential flotation of ores are (1) the sepa-

ration of lead sulphide from zinc sulphide, and (2) the separation of zinc and copper bearing sulphides from sulphides of iron.

Methods seeking separation of one sulphide from another by flotation are of two classes, viz.: (1) those in which a difference in flotability of the two minerals is created or enhanced by conscious control of one or more of the activating phenomena of the process, and (2) those in which the chemical nature of one of the sulphides is changed, with a resulting change in the nature of the surface. Methods of the first class truly accomplish "differential flotation" and this denomination should be closely restricted to such methods. Methods of the second class, by preferential chemical action, so change one of the sulphide minerals that it never floats, leaving the unaltered mineral to float under proper conditions. Freeman (Min. and Sci. Press, June 5, 1920, abstract from Proc. Australasian Inst. of M. and M.) calls this "preferential flotation" and this terminology may profitably be adopted.

Differential flotation of galena in the presence of blende may be brought about, in pulp-body concentration, by proper control of gasification and of kind and quantity of "oil," in the presence of fresh water. This merely means that as the conditions under which gas is precipitated from the water are gradually brought up to the point where maximum precipitation occurs, a point is first reached where the gas precipitates preferentially on the galena. Gas precipitation is increased by agitation, by acid, and by heat, and the necessary adherence of the gas bubbles after precipitation is aided by the presence of oil on the sulphide particles. While there is no direct evidence on the point, there is indirect evidence pointing to the fact that oiling of galena takes place under given conditions more rapidly than corresponding oiling of blende. Hence by using a very small amount of oil with, in the agitation-froth process, a relatively gentle agitation, differential oiling of the galena probably occurs. And with the same relatively gentle agitation and in the cold, differential precipitation of gas on the oiled galena occurs, resulting in flotation of galena and non-flotation of the blende. Increase in agitation or application of heat or addition of acid or two or all of

these accompanied by the addition of more oil, either of the same or a different kind, will result in raising blende.

The differential flotation effect above noted and described is greatly enhanced by the presence, in acid solution, of certain salts, chiefly sulphates; or by sulphur dioxide (SO_2) in an otherwise non-acid pulp; or, it is said, by the presence of sulphur in solution; or by a permanganate; or soda ash. When sulphates are used in the flotation of the galena, subsequent flotation of the blende is brought about by a marked raise in temperature and by the addition of more acid and oil; where SO_2 is used it is driven off after the galena flotation by heat or aëration or is chemically neutralized, and acid and more oil are added; with "sulphur in solution" and with permanganate, oil, acid and heat are added to accomplish the blende flotation; with soda ash more oil and a salt, such as copper sulphate, produce the blende flotation.

It is distinctly in favor of the methods that float galena away from blende that, since associated silver usually floats with whichever mineral is first floated, silver in a lead concentrate is much more valuable than silver in a zinc concentrate.

"Preferential flotation" of blende from a mixture of blende and galena is accomplished by altering the galena through chemical reaction. This has been done by a low-temperature oxidizing roast, which changes the galena, at least superficially, into the non-lustrous sulphate; or by digesting the feed with a hot acid solution of ferric chloride; or with bichromates; or with "acid salts."

It is to be especially noticed in connection with these processes that, since the altered galena is non-floatable, if separation of lead from gangue is to be effected, a mixed float of lead and zinc minerals must first be made and this concentrate then be chemically treated and again subjected to flotation to separate the blende. This treatment leaves a lead concentrate as the underflow from the flotation machine, where otherwise a worthless leady-tailing would be discharged.

Separation of copper-bearing sulphide from iron sulphide has been worked out on a laboratory scale by using a deficient amount

of a flotation agent, which, when present in greater amount, will float both of the sulphides, although even under such circumstances exhibiting some differential action. No one flotation agent possesses this differential characteristic. Each such case is, in the present state of our knowledge, an individual one which must be investigated, having in mind the cardinal principle.

Sampling. In laboratory work sampling is simple, if the batch to be tested is small and ground dry. In such case an assay sample of the feed should be riffled out of the lot, and the products of each test should be dried and weighed and samples for assay then taken by riffling. With a known weight of feed a check is thus obtained on the accuracy and care of the testing work and on the accuracy of the sampling and assaying.

When the sample for laboratory testing is a wet pulp, sampling the feed is a difficult matter. Practice in the writer's laboratory is to get the solids in suspension as completely as possible by stirring and then to take duplicate samples by dipping out two 300-cc. beakerfuls of pulp, plunging the beaker quickly well beneath the surface of the pulp. These samples are then dried, riffled and assayed. Concordant results are taken to indicate accuracy.

In a test mill the sampling comprises both tonnage and assay samples. Sampling of feed, tailing and concentrate for assay should be automatic. Feed samples will be more nearly representative the finer ground and more fluid the pulp sampled. Hence where possible the material sampled should be the actual flotation feed just before it enters the flotation machine. Provision should be made at this point also for tonnage sampling. This may be by volume or weight as desired, either method requiring the simultaneous taking of a moisture or pulp-density sample. For tonnage measurements by volume a container, large enough, if possible, to take the total flow for a minute, should be provided, and the launder carrying the pulp to be sampled should be so arranged that the whole flow can be quickly diverted into the container and subsequently as quickly away. If oil is present in the pulp to be sampled the container should be calibrated to overflow, as the froth which forms on

the pulp in the container would mask any other end point. When the tonnage sample is weighed, the container should be set on a pair of platform scales, the requirements as to size of container and pulp diversion being the same as before. Now, however, no requirement as to overflowing obtains.

Pulp densities are obtained from the determination of the weight of a small sample of known volume. This should be cut from the sample stream at the time that the tonnage sample is being taken, by means of a slotted cutter of the type and proportions shown in Fig. 17, and should be washed into a tared,

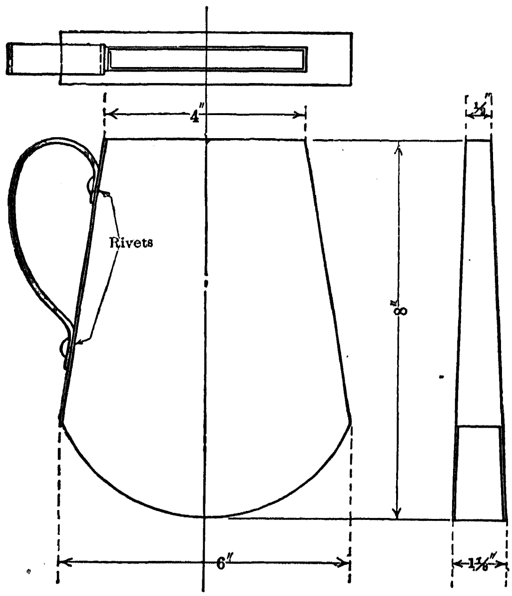


FIG. 17
Sample cutter

wide-mouthed measuring container by means of a measured volume of water. Then, knowing the specific gravity of the ore, the percentage of solids and hence the weight of solids in the sample can be computed by formula (see page 163) or can be obtained from the chart, Fig. 18.

Preparation of ores for tests. The preparation of ores for flotation tests depends upon the kind and purpose of the test

to be made. In general the common sense of the operator is a sufficient guide. Thus, if the test is a laboratory test made

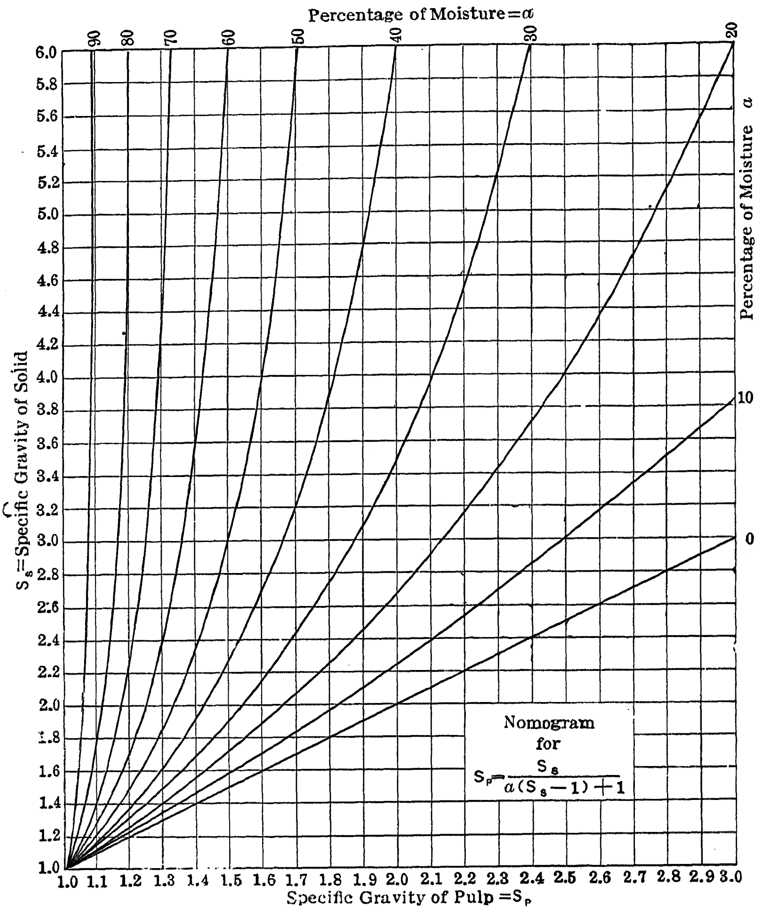


FIG. 18
Nomogram for determination of pulp density

to determine the reason for some mill performance, the feed pulp will be prepared in the mill itself and the sample will be taken, of course, at the head of the mill flotation machine. But if the test is a laboratory test on an ore to determine, for instance, the amenability of the ore for flotation, it is, in general, best to grind the ore dry for all of the preliminary testing work.

The usual machines employed are a small crusher to bring the ore to, say, minus-$\frac{1}{4}$ or minus-$\frac{3}{16}$-inch; a pulverizer to further reduce it to pass, say, a one-millimeter screen; and finally a laboratory pebble mill. Such a method of reduction produces a pulp somewhat different from that produced by ordinary wet-grinding mill methods. If all the pulp is dry-ground through some limiting screen, a greater proportion will pass a 0.074-mm. screen than would be the case with a wet-ground pulp in the mill where the same limiting upper size is sought. This difference will tend to cause the results with the dry-ground ore to be better than would be obtained on the mill pulp. On the other hand, if the mill grinding is to be done in the presence of the flotation agent, the favorable effects of this treatment on subsequent flotation will probably more than offset the advantage to the laboratory test of the finer grinding. It may be set down also, as a general tendency, that, all other conditions being as nearly as possible equal, finer grinding is necessary for a laboratory test than for mill operations, if the same result is to be obtained.

In general, as regards the storage of feeds for testing purposes: Selection in flotation depends entirely upon the nature of the surfaces of the solid particles in the pulp. It is probably not strictly accurate to say that the surfaces of the selected solids must be clean and of a truly metallic, adamantine or resinous luster, but the behavior of samples under certain conditions in flotation testing indicates that some such requirement does exist. It is a fact that a sulphide ore, which has been so treated as to induce oxidation of the sulphide minerals or coating of these minerals with any sort of a solid or semi-solid covering is entirely changed insofar as flotation character is concerned; thus a sample of pulp cut at the head of a machine in which successful flotation is being carried forward, may, and usually will, if allowed to stand for a considerable length of time, or if evaporated to dryness and again wet, partially or utterly fail to give any satisfactory concentration by flotation, if treated under conditions in the laboratory machine otherwise similar to conditions in the mill. Similarly a dry sample that is allowed to stand around a laboratory in which there is any consider-

able amount of acid fumes, may, after a considerable length of time, show entirely different flotation characteristics from those exhibited by the fresh sample. Ores ground to a given size in a pulverizer of the Braun disk type, may show an entirely different behavior in a flotation test from another sample from the same lot, ground dry in a laboratory pebble mill to pass the same limiting screen, and the dry-ground samples in both cases may show an entirely different flotation performance from one wet-ground in a laboratory pebble mill. An ore ground in fresh water in the laboratory pebble mill may show a different flotation result from one ground in the presence of oil in the same mill, but the effect here is largely one of the dispersion of the oil through the pulp and it appears that if, following grinding in the presence of fresh water, the pulp is subjected to sufficient premixing with the oil to cause a dispersion of the oil as thorough as that in the case of grinding in the presence of the oil, the flotation result will be the same.

Accessory tests. Any testing for flotation, upon the results of which a mill design is to be based, should be accompanied by tests as to the behavior of the ore in grinding machines, mechanical classifiers and settlers, and on the concentrate to determine the ease with which it may be sunk and the extent to which it can be dewatered by settling and thickening. No finished design on anything more than test-mill scale should be embarked upon until such tests have been carried out in actual machines. But for preliminary work much information will be afforded to an experienced engineer by the following tests:

Grinding. In a bomb of the dimensions and design illustrated in Fig. 19, imbed a No. 6 detonator with 100 gm. of a sand sized between 1.168 mm. and 0.833 mm., the sand having been obtained from a rock on which the performance of a given grinding mill is known. Explode the detonator, remove and size the sand. Run a duplicate test to insure the constancy of the exploding force of the detonator. Present-day detonators in good condition will give results that check within the limits of error of a screen test, i.e., the curves representing the sizing analyses will be practically coincident. Make duplicate tests in

the same way on the ore under investigation. If the results of the duplicates again check, the force exerted by the four detona-

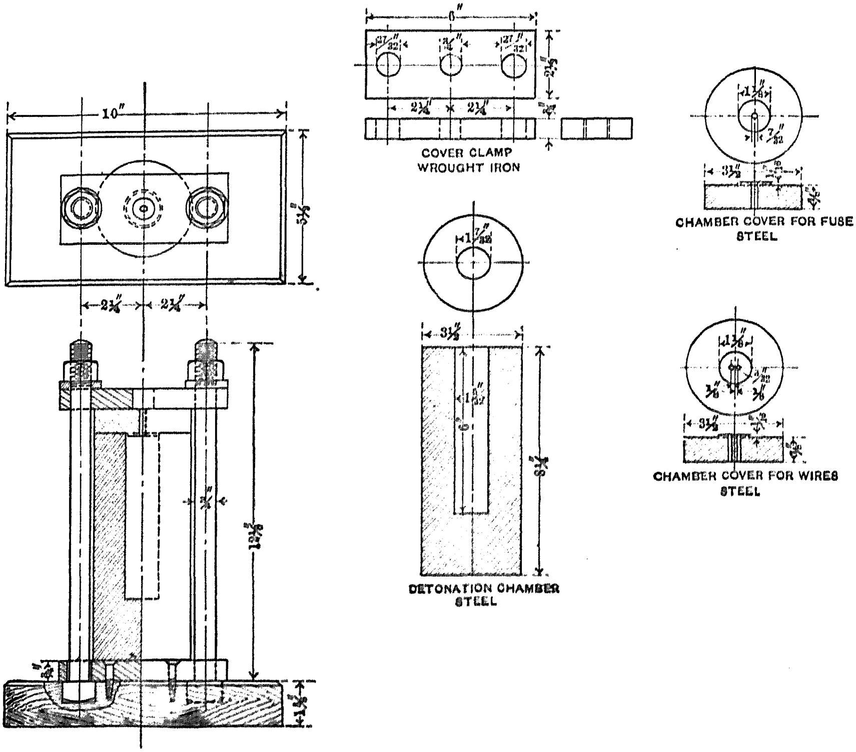


FIG. 19
Electric detonator apparatus for crushing tests, U. S. Bureau of Mines

tors may be concluded to have been constant and the results may then be compared directly.

Classifying and settling performances of an ore may be predicted from a comparison of the performance of samples with the behavior of like samples of a known ore whose mill performance is known. Thus artificial samples of the known and unknown ore made up to the same screen analysis should be stirred up in a beaker and the beaker then be allowed to stand for, say, one minute, at the end of which time the material in suspension should be poured into another beaker where it is again allowed

to settle for a minute or longer as desired and the suspended solids again poured off. This manipulation should be repeated until enough points on a settling-rate curve are established so that this curve for the known and unknown ores can be compared. Knowing then the performance of given apparatus on the known ore, the performance of the same apparatus on the unknown ore can be predicted.

Concentrate handling consists in breaking down froth concentrate, thickening the same by settling, and filtering the thickened product. Sampling and transporting from thickener to smelter also offer problems, but these are not part of the subject matter under present discussion. A few general principles are all that can be set forth here to aid the experimenter

Froth may be broken down by impact or by the force of surface tension or both. Unfortunately the same forces also tend to make froth. Hence it is essential that they be utilized in a different way or to a different extent when the end in view is froth destruction. If a small amount of froth is placed on a body of fresh water or water but slightly contaminated with a frothing agent, the tension of the water surface will pull the froth mass apart into individual bubbles and will then so extend most of the individual bubbles, especially the larger ones, that the films will rupture and the solid load will sink. The bubble film may also be ruptured by piercing or puncturing. In practice this is accomplished by directing a spray of water upon the froth. A froth is a system in more or less unstable dynamic equilibrium under the forces of gravity, surface tension and viscosity. Anything that tends suddenly to upset the equilibrium of the system will tend to break down the froth. A sudden change in surface tension can be brought about by spraying with a substance or solution whose surface tension is different from that of the bubble films.

The three phenomena outlined in the last paragraph are all utilized in froth breaking. General practice in the mills is to run the froth concentrate through launders to Dorr tanks fitted with a peripheral curb to prevent froth overflow, and to spray the surface of the tanks, particularly near the center, usually

with fresh water, in order to puncture the bubble films. Occasionally the water used is contaminated with a substance which markedly lowers the surface tension. This upsets the equilibrium of the forces acting in the bubble films, in addition to the puncturing effect. This latter procedure is necessary only in the case of obstinately persistent froths.

Froths carrying a high percentage of solids are more persistent than those with a low percentage and more elaborate froth breaking equipment is necessary for their treatment. Such froths result from ores carrying a high percentage of mineral or high percentages of kaolinized matter. They result also from agitation methods of froth formation as differentiated from pneumatic methods.

Certain flotation agents, notably petroleum products and wood-tar oils produce persistent froths. Also the froths produced with more than 1 per cent. of oil on the ore are harder to break down than those produced with smaller quantities.

Pilot machines should form a part of every mill installation, especially in its early life. The pilot machine should give a quick and easily visible indication of the character of the plant tailing. Either a gravity concentrator or a pneumatic flotation machine may be used. In either case a portion of the general tailing of the flotation plant is diverted to the pilot machine and there treated slowly in order that all of the recoverable mineral may be exposed for inspection. Probably the best pilot installation is a combination of the two machines, in which the gravity concentrator will show the coarse mineral that is being lost and thus guide grinding operations, while the pneumatic flotation machine will bring to view the finer mineral in the tailing.

Oil Testing

The usual purpose of oil testing in flotation work is to determine that the oil in question is similar in physical properties and therefore probably in its behavior in the flotation cell to a given prior shipment, or in order to set specifications for flotation oil purchase. The following tests will give such information.

Color. Color is best tested by comparison in containers of the same size and shape with a sample of the oil with which comparison is to be made. In case no reference sample is available, a general specification as to color, without reference to a color chart, is all that is justifiable.

Limpid point. Take a sample of about 10 cc. in a thick-walled test tube. Cool gradually, at the same time stirring constantly, and lift the oil with a thermometer in such a way as to allow it to flow down the sides of the test tube. Cooling should be continued until crystals can be seen in the oil as it flows down the side of the tube. Note and record temperature at which crystallization starts, and also that at which complete solidification occurs on further cooling. Now warm gradually, stirring constantly as before, until crystals disappear and note temperature of disappearance.

Specific gravity. Place sample of oil in a cylinder 1 to $1\frac{1}{2}$ in. diameter and about 6 in. deep. Bring to a temperature of 15° C. Float a hydrometer in the oil, taking care that it does not strike the sides of the cylinder, and read the specific gravity of the sample from the hydrometer scale. Repeat the determination at temperatures of 25 and 60 degrees.

The following method should be followed in the determination of specific gravity by means of a specific gravity bottle. In ordinary work a 25-cc. bottle will be satisfactory. Clean the bottle thoroughly with chromic acid solution, rinse with water and dry. Weigh dry bottle to $\frac{1}{10}$ mg. Fill with distilled water at 15° C. Clean and dry the outside of the bottle thoroughly and again weigh with the same degree of accuracy. Next dry the bottle and fill with oil, bring to the same temperature as that of the water in the previous weighing, clean and dry the outside of the bottle and weigh. The weight of oil divided by the weight of water gives the relative specific gravity of the oil compared with water at that temperature. Repeat at 25° and 60° C.

Viscosity. The viscosity of a liquid is the transient resistance offered by the liquid to deformation. The coefficient of viscosity is the ratio of the shearing stress during such deformation to the rate of shear, expressed in proper units, and is a constant

for any one fluid under given conditions of temperature and pressure. The "specific viscosity" is the ratio of the viscosity of the fluid under consideration to the viscosity of water at a specified temperature. The latter value is the one which is of interest in testing and classification of flotation oils and the temperatures of principal interest to the flotation operator are those at which the oil is received, at which it is stored, at which it is to be transported from storage to the flotation cell, and that of the flotation pulp in which it is to be used. The apparatus employed in the determination varies according to the viscosity of the oil under investigation. For relatively viscous oils, some form of the Engler viscosimeter is ordinarily employed. For relatively mobile oils, the Ostwald apparatus is used.

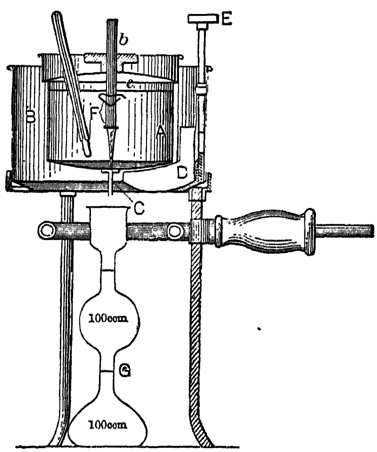


FIG. 20
Engler viscosimeter

The Engler viscosimeter is illustrated in Fig. 20. It consists essentially of a cup (*A*), fitted with a pipe orifice (*C*), a container (*B*) with a stirring mechanism (*E*, *D*), and a graduated flask (*G*). The cup (*A*) is provided with an air insulated cover (*c*) and a pointed wooden stopper (*b*).

The following is the procedure for the determination of the specific viscosity of an oil with this apparatus. Take a sample approximately 15 per cent. greater in bulk than required to fill the cup (*A*) to the top of the gauge points. Strain the sample through a 100-mesh wire screen to remove foreign matter. Clean the cup (*A*) with solvents such as benzol or alcohol in order to remove any oily substances, taking particular care that the discharge orifice is thoroughly clean and dry. Soft tissue paper or filter paper is best for this cleaning process. Now assemble the apparatus, placing the pointed wooden stopper in the orifice and fill the inner cup to the top of the gauge points with the oil

to be tested. Place cover (*c*) in position and insert thermometer (*F*). Fill chamber (*B*) with an oil of higher boiling point, usually a heavy lubricating oil, and insert a thermometer in this liquid. Heat slowly by means of the burner (*H*), stirring the heating liquid by means of the stirrer (*E*, *D*) until the desired temperature is reached. Hold the temperature at this point until the oil in cup (*A*) has reached the same temperature. Place container (*G*) in such a position under the orifice that the oil will flow down the side of the container and thus prevent the formation of a froth. Lift the pointed wood stopper and determine with a stop watch the elapsed time in filling the flask to the graduation mark. Repeat until concordant results are obtained. The time required to discharge an equal volume of distilled water under the same conditions of temperature and head on the discharge orifice should now be determined, first cleaning the inner cup thoroughly with oil solvents. The ratio of the time of outflow of the oil to that of the water is taken as the specific viscosity of the oil, referred to water at the same temperature, and is called the "Engler degree."

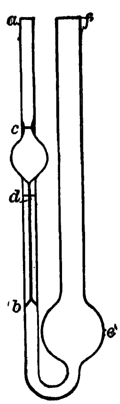


FIG. 21
Ostwald viscosimeter

The Ostwald viscosimeter used for mobile liquids is shown in Fig. 21. It consists of a fine capillary tube (*db*) about 10 cm. long and 0.4 mm. bore, connected at the upper end to a bulb containing a definite volume between the graduation marks (*c*) and (*d*). The capillary is connected at the lower end by a U-tube to a large bulb (*e*), which in turn is connected to a tube of convenient size for filling. Before using, the apparatus should be cleaned first with suitable oil solvents such as benzol, acetone, ether, or alcohol and then with warm chromic acid solution, which latter should be allowed to stand in the apparatus for an hour or more. Finally it should be washed out with water and dried thoroughly by means of a current of air. To make a determination of specific viscosity, introduce into the leg (*f*) by means

of a calibrated pipette, a definite volume of oil sufficient to cause the bulb (*e*) to stand two-thirds full. Place the viscosimeter in a water bath maintained at a definite constant temperature and allow it to stand therein until the oil and apparatus are of the same temperature as the bath. The time required will range from 30 minutes to 1 hour, according to the difference in temperature between the bath and the surrounding atmosphere. Next place a rubber tube at (*a*) and suck oil into this limb to some point above the mark (*c*). Remove the tube and allow the oil to flow down through the capillary (*db*) by gravity, taking the elapsed time by stop watch for the meniscus to pass from graduation (*c*) to graduation (*d*). This operation should be repeated until concordant results are obtained. A variation in elapsed time of 0.1 to 0.5 per cent. is allowable. Repeat the above described procedure with water at the same temperature, using the same volume of water as of oil. The ratio of time of flow of oil multiplied by the specific gravity of the oil, to the time of flow of the water multiplied by the specific gravity of the water gives the specific viscosity of the oil compared with water at the given temperature.

Tar acid determination should be made on all tars and tar derivatives. The method of procedure is as follows: Place 100 cc. of oil in the separatory funnel shown in Fig. 22. Add from 30 to 50 cc. of sodium hydroxide solution (sp. gr. 1.1) and mix by turning the funnel end for end several times. Allow the liquids to stratify roughly, and draw off the lower layer. Repeat this operation with new 30- to 50-cc. portions of sodium hydroxide solution until the aqueous layer, on stratifying, shows no further coloration. Read the volume of residual oil in the separatory funnel. The decrease in volume gives the volumetric percentage of tar acids extracted. Now wash the combined alkaline extracts with C. P. benzol in the separatory funnel, allow the mixture to stratify, and draw off the aqueous layer. Heat this portion on a steam bath until the odor of benzol has disappeared. Cool, place in the separatory funnel and liberate the tar acids with 20 per cent. sulphuric acid solution. Allow to stratify and read off the volume of tar acids liberated.

With some oils preliminary distillation is necessary in order to later obtain separation of the alkaline solution from the oil. In such cases distill 100 cc. of the oil, continuing the distillation until at least 95 per cent. by volume of the oil has distilled over or until a temperature of 360° C. has been reached. Read the volume of the distilled oil and determine the tar acids on this portion by the procedure above outlined. Calculate on basis of original bulk.

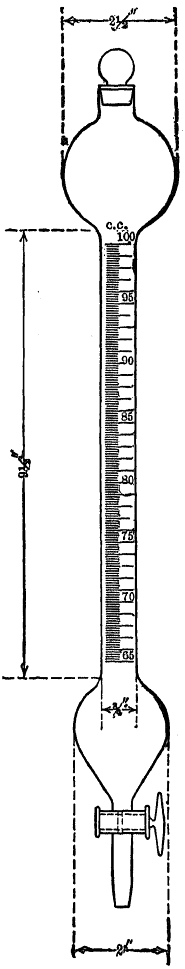


FIG. 22
Tar acid separatory funnel

Sulphonation is used for the purpose of determining unsaturated and aromatic constituents in an oil. Its chief use is for determining adulterants, such as mineral oils, in pine and tar oils. The test is carried out by placing 2 cc. (accurately measured) of the oil in a Babcock cream-testing bottle and treating the oil with 37-normal sulphuric acid in small quantities, shaking the oil and acid mixture thoroughly after each addition. When no more heat is developed on the addition of acid and when about 40 cc. have been added, heat for one hour at a temperature of 98° to 100° C. The bottle may then be filled to the upper graduation with sulphuric acid (sp. gr. 1.84). The mixture should now be centrifuged for five minutes and the volume of residual oil read from the graduations. The volume of residual oil divided by the original volume represents the percentage of unsulphonated material present in the original oil. This volume should be not over 3 or 4 per cent. for a good steam-distilled pine oil. For tars, the volume will vary considerably.

Distillation test. The purpose of the distillation test is to determine the percentages of the constituents of different boiling points. The apparatus for such a test is shown in Fig. 23.

Place 500 gm. of the oil to be tested in the flask (*a*), connect with condenser tube (*c*) by means of a cork stopper, insert thermometer (*b*) in such a position that the top of the mercury bulb

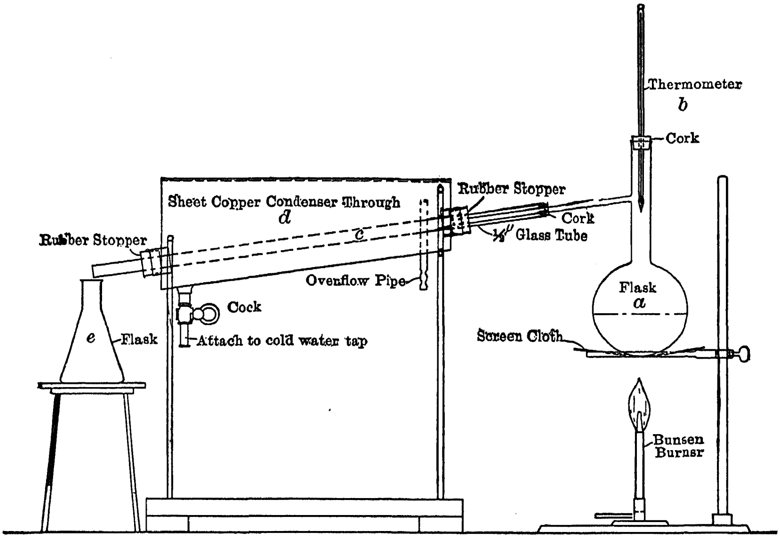


FIG. 23
Distillation apparatus

is on a level with the bottom of the side tube in the distilling flask, taking care that the bulb is in the center of the neck of the flask. Fill the condenser trough (*d*) with water. Place a weighed receiving flask (*e*) at the outlet of the condenser tube. Heat the flask (*a*) cautiously until such time as the water is completely removed from the sample. This time is indicated when bumping stops and when there is gentle ebullition of the boiling liquid. After the water is removed heat at such a rate that approximately two drops per second will fall from the condenser tip. The receiving flask should be changed at a temperature of 150° C. and at the end of each succeeding 50-degree rise. When the distilling temperature is above 250° C., the water in the condenser trough should be warmed to prevent solidification of the condensing material. Carry the distillation to a temperature of 350° C.

The interval of 50° C. in the end points of the various fractions is satisfactory for general rough distillation. If it is suspected that the oil is a particular substance, and a close analysis by distillation is desired, the points in the distilling scale will be chosen at intervals dependent upon the substance under investigation. For instance, with a tar or tar oil, the points will correspond to similar points in the manufacture of these oils. These points will be in general determined by specifications under which the oil is sold. Typical distillation analyses of common flotation oils are presented in Tables I to X.

TABLE I

DISTILLATION ANALYSES OF PINE OIL

Temperature ranges, degrees C.	Percentages					
	1	2	3	4	5	6
Below 150	1.75	3.25	3.00	2.60	3.00	1.75
150–200	39.50	10.75	40.75	56.00	38.35	68.50
200–250	56.00	83.75	54.25	40.30	48.25	29.25
Residue	1.75	(b) 1.25	(c) 1.00	(a) (b) 1.10	9.50	(a) 0.50
Loss	1.00	1.00	1.00	......	1.00	......
Total	100.00	100.00	100.00	100.00	100.00	100.00
Refractive index	1.4910	1.4780 20° C.	1.4820 20° C.	......	1.4860 20° C.	1.4925 20° C.

1. C. R. Hadley & Co., San Francisco, Cal.
2. General Naval Stores Co., No. 5 Pine Oil.

3 and 5. Yaryan Standard Pine Oil.
4. Yaryan Naval Stores Co., Brunswick, Ga.
6. Rosin and Turpentine Export Co.

(a) Includes loss. (b) Residue above 225° C (c) Residue above 228° C

TABLE II

DISTILLATION ANALYSES OF WOOD TAR

Temperature ranges, degrees C.	Percentages	
	1	2
Below 150	....	....
150–200	24.50	33.30 (c)
200–250	18.70	
250–300	21.30	(d) 33.30
300–350	30.00	(e) 8.3
Residue	(a) (b) 5.50	(a) (f) 25.10
Loss	....	....
Total	100.00	100.00

1. Pensacola Tar and Turpentine Co. 2. Origin unknown.

(a) Includes loss.
(b) Residue above 330° C.
(c) Fraction 150°–250° C.
(d) Fraction 250°–340° C.
(e) Fraction 340°–343° C.
(f) Residue above 343° C

TABLE III

Distillation Analyses of Crude Wood Oil

Temperature ranges, degrees C.	Percentages						
	1	2	3	4	5	6	7
Below 150	8.50	1.00	23.00	2.50	7.50	3.50	11.50
150–200	25.50	46.25	5.50	69.00	13.00	1.75	8.50
200–250	11.00	13.50	9.50	11.75	18.00	25.50	23.00
Residue	(a) 55.00	38.25	61.00	15.75	60.50	68.25	(a) 57.00
Loss	.....	1.00	1.00	1.00	1.00	1.00	....
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Refractive index	.....	1.5112 20° C.	.....	1.4910 20° C.	1.5280 20° C.	....	....

1. Pensacola Tar and Turpentine Co.
2. Yaryan Naval Stores Co.
3. United Naval Stores Co.
4. Geo. P. Jones Co., St. Louis, Mo.
5. Georgia Pine Turpentine Co., N. Y.
6. American Turpentine and Tar Co.
7. Florida Wood Products Co.

(a) Includes loss.

TABLE IV

Distillation Analyses of Wood Creosote

Temperature ranges, degrees C.	Percentages					
	1	2	3	4	5	6
Below 150	4.00	3.00	10.00	10.00	6.75	9.00
150–200	3.25	21.50	7.00	6.00	5.75	10.50
200–250	58.50	47.50	62.00	37.50	19.50	13.75
250–300	......	......	......	......	......	......
Residue	(a) (b) 34.25	(b) 27.00	(b) 20.00	(b) 45.50	(b) 67.00	65.75
Loss	......	1.00	1.00	1.00	1.00	1.00
Total	100.00	100.00	100.00	100.00	100.00	100.00
Refractive index	......	......	......	......	1.5370 20° C.	.....

1, 2. Pensacola Tar and Turpentine Co.
3, 4. Cleveland Cliffs Iron Co.
5. Geo. P. Jones Co., St. Louis, Mo.
6. Georgia Tar and Turpentine Co.

(a) Includes loss. (b) Residue above 250° C.

TABLE V

Distillation Analyses of Crude Coal Tars

Temperature ranges, degrees C.	Name of distillate group	Percentages				
		1	2	3	4	5
Below 150	Water and ammoniacal liquor	8.0	6.0	3.5	23.0	4.0
150–200	Benzol	3.0	4.5	1.0	2.0	4.0
200–250	Naphthalene	17.0	24.0	22.0	14.0	11.0
250–270	Creosote	6.0	3.5	5.0	3.0	6.0
270–350	Anthracene	22.5	9.0	15.0 (a)	15.0	25.0
Residue	Pitch	42.5	52.0	53.5	42.0	49.5
Loss	.....	1.0	1.0	....	1.0	0.5
Total	.....	100.0	100.0	100.0	100.0	100.0
Tar acid	.....	18.0	....	....	18.0	18.0

1, 4. Barrett Manufacturing Co.
2. C. G. Betts & Co.
3. American Tar Products Co.
5. Denver Gas and Electric Co.

(a) Includes loss.

TABLE VI

Distillation Analyses of Coal–Tar Oil

Temperature ranges, degrees C.	Name of distillate group	Percentages				
		1	2	3	4	5
Below 150	Ammoniacal liquors	0.50	1.00	12.50	2.25	6.50
150–200	Benzol	9.50	5.50	39.50	2.50	60.00
200–250	Naphthalene	49.50	47.00	38.00	30.00	29.50
250–270	Creosote	11.75	15.50	....	7.50	1.00
270–350	Anthracene	24.50	28.50 (a)	.... (a)	28.50	2.50 (b) (a)
Residue	Pitch	3.25	2.50	10.00	28.25	0.50
Loss	.....	1.00	....	....	1.00	....
Total	.....	100.00	100.00	100.00	100.00	100.00
Tar acids	.....	9.58	6.50	....	2.00	24.00

1. Coal tar oil (heavy) from American Tar Products Co.
2. Barrett Manufacturing Co.
3. Light oil from Barrett Manufacturing Co.
4. Tar oil from American Tar Products Co.
5. American Tar Products Co.

(a) Includes loss.
(b) Above 300° C.

TABLE VII

Distillation Analyses of Coal-tar Creosote

Temperature ranges, degrees C.	Name of distillate group	Percentages 1	2	3	4	5	6
Below 150	Water and ammoniacal liquor........	2.0	2.50	2.50	1.00	3.00	9.75
150–200....	Benzol............	18.0	23.50	28.00	6.50	5.50	20.25
200–250....	Naphthalene........	60.0	58.00	53.00	40.00	50.50	42.75
250–270....	Creosote............	7.0	4.75	5.50	30.00	15.50	12.75
270–350....	Anthracene.........	9.0	5.00	6.50	19.50	18.50	9.25
Residue...	Pitch...............	3.0	5.25	3.50	2.00	6.00	5.25
Loss.......	....	1.0	1.00	1.00	1.00	1.00	....
Total....	....	100.00	100.00	100.00	100.00	100.00	100.00
Tar acids..	....	25.0	32.0	35.00	1.00	7.00	10.00

1, 2, 3. Barrett Manufacturing Co.
4. Creosote oil from Stimpson Equipment Co.
5. American creosote No. 2 from American Creosote Works.
6. F. J. Lewis Co.

TABLE VIII

Distillation Analyses of Stove Oil

Temperature ranges, degrees C.	Name of distillate group	Percentages 1	2	3	4	5	6
Below 150 .	Gasoline............	None	None	1.75	9.50	None	None
150–200....	Kerosene...........	30.50	10.00	62.50	22.00	None	3.50
200–250....	Heavy illuminating.	55.75	26.50	29.25	9.80	19.00	45.00
250–300....	Light lubricating...	9.00	43.25	None	9.50	44.25	38.25
300–350....	Heavy lubricating..	None (a)	17.00 (a)	None (a)	46.50 (a)	24.50	11.75 (a)
Residue...	....	4.75	3.25	6.50	2.70	11.25	1.50
Loss.......	....	....	....	....	....	1.00	....
Total....	....	100.00	100.00	100.00	100.00	100.00	100.00

1, 2, 3, 4. No. 9 oil from Utah Oil Refining Co. 5. Stove oil from Standard Oil Co.
6. Stove oil from Union Oil Co., Los Angeles, Cal.

(a) Includes loss.

TABLE IX

Distillation Analyses of Asphaltum-base Residuum

Temperature ranges, degrees C.	Name of distillate group	Percentages					
		1	2	3	4	5	6
Below 150	Gasoline	0.50	0.50	0.50	0.23	None	None
150–200	Kerosene	1.00	2.00	1.50	0.25	None	None
200–250	Heavy illuminating	6.00	5.75	7.50	4.40	7.50	1.50
250–300	Light lubricating	10.00	12.20	12.50	16.66	13.50	7.00
300–350	Heavy lubricating	33.50	26.50	19.25	58.00 (a)	64.50 (a)	20.50
Residue	Asphaltum base	48.00	52.05	57.75	20.46	14.50	70.00
Loss	....	1.00	1.00	1.00	....	....	1.00
Total	....	100.00	100.00	100.00	100.00	100.00	100.00

1, 2. Smelter fuel oil from Standard Oil Co., Lyoth, Cal.
3. Smelter fuel oil from Garfield Smelting Co.
4. California fuel from California Oil Fields.
5. Wayside residuum from Standard Asphalt and Rubber Co.
6. Jones oil, from Geo. P. Jones, Coffeyville, Kan.

(a) Includes loss.

TABLE X

Distillation Analyses of Paraffin-base Residuum

Temperature ranges, degrees C.	Name of distillate group	Percentages				
		1	2	3	4	5
Below 150	Gasoline	None	None	None	7.50	18.00
150–200	Kerosene	None	None	None	16.00	14.50
200–250	Heavy illuminating	None	3.30	14.25	13.75	13.00
250–300	Light lubricating	7.66	5.00	36.00	13.75	13.50
300–350	Heavy lubricating	79.65	69.00	39.75	20.00	18.50
Residue	Heavy paraffin with small amount of asphaltum and free carbon	12.69 (b) (a)	22.70 (a)	10.00 (a)	29.00 (a)	22.50 (a)
Loss	....	....	....	....	....	....
Total	....	100.00	100.00	100.00	100.00	100.00

1. Electra residuum from Utah Oil Refining Co.
2. Spring Valley cylinder stock from Utah Oil Refining Co.
3. Texas crude oil from Texas Oil Co.
4. Eldorado, Kansas crude oil from C. O. Kimmell, Bartlesville, Okla.
5. Paraffin base crude oil from Geo. P. Jones Co.

(a) Includes loss. (b) Residue above 340° C.

It should be borne in mind, as is shown by the tables, that a distillation analysis of one member of a given class of oils is not closely representative of all members of the class and that in flotation an oil of a given class may depart far from the distillation analyses herewith given, and yet serve its purpose in a wholly satisfactory manner. However, if a given oil is being bought regularly from a given manufacturer and the oil mixture in a flotation plant has been compounded and adapted to the needs of the plant, then any marked variation in distillation analysis is likely to be accompanied by a noticeable variation in the flotation results obtained by the oil mixture. It is possible, however, for a manufacturer to furnish oils of considerably different chemical constitution which will give distillation analyses that check within the allowable limits established by practice, and where flotation results in a plant change with the initial use of a new lot of oil in the oil mixture, the composition of the oil should be investigated beyond the point possible by distillation analysis alone.

Refractive index. The velocity of light waves is different in different media. This fact causes light rays passing from one medium into another to be bent. The amount of bending resulting from the passage of a ray of light from air into a given substance is a specific property of the substance. It is expressed by the ratio between the sine of the angle of incidence and the sine of the angle of refraction. The angle of incidence is that between the incident ray and the perpendicular to the surface of the substance at the point of incidence. The angle of refraction is that between the same perpendicular and the direction of the ray in the substance. The index of refraction thus measured will be greater than 1.0 for all solid or liquid substances.

The index of refraction is determined by means of an instrument called a refractometer. The instrument is built in several different forms. The Abbé, which is a common form, consists of two prisms of dense flint glass mounted in water-jacketed frames for temperature control. These prisms are movable around an axis at right angles to the axis of the instrument telescope and are rigidly attached to a pointer which travels

over a scale attached to the telescope. The pointer thus measures the angular relation between the prism faces and the telescope axis. Light entering the lower prism is in part totally reflected and in part refracted into the telescope. When the prisms are viewed through the telescope, therefore, a part of the field is dark and a part light. When the line of division between these two parts of the field is brought into coincidence with the junction of the cross-hairs of the telescope the pointer, in the usual instrument, reads directly the index of refraction of the liquid referred to air, at the temperature of the investigation. An explanation of the optics of the instrument is to be found in any good book on general physics.

Before determining the index of refraction of an oil, water should be passed through the circulating system and the plates brought to the temperature at which the determination is to be made. This temperature should be in the neighborhood of 25° C., but may be varied according to the consistency of the material under examination. When the desired temperature has been reached, clean the plates thoroughly with soft paper and ether or other suitable oil solvent. Place a drop of the oil to be examined on the horizontal plate, close the plates and lock by means of the locking screw. Allow time for the oil to attain the temperature of the plates. Now move the plates backward and forward across the line of sight of the telescope until the point is reached at which the division between light and shadow is coincident with the intersection of the cross-hairs. Read from the vernier the index of refraction of the oil. The determination should be repeated until concordant results are obtained. Read the temperature again and clean the plates with cotton and ether before setting the instrument aside.

The refractive indices of several typical flotation oils are given in Table XI.

Odor is an important aid in the identification of an oil but characteristic odors cannot be satisfactorily described except against a background of experience. The experimenter should acquaint himself with the odors of the common classes of flotation oils as given in the list in Table XI.

Fluorescence is characteristic of petroleum products and will serve to indicate the presence of any considerable quantity of such products in mixtures.

Table XI presents certain physical properties of samples of some of the more prominent flotation oils. It will be seen that these properties vary markedly. Shipments of oil under the same name from the same dealer will vary considerably and shipments of supposedly similar oils from different dealers will vary widely. No definite relation between physical properties and usefulness in flotation has yet been worked out.

TABLE XI

PHYSICAL PROPERTIES OF FLOTATION OILS

Kind of oil	Color	Odor	Consistency or viscosity	Density range	Refractive index	Tar acids, per cent.
Paraffin-base residuum	Greenish, almost transparent, to black	Odor of illuminating oils; often disagreeable, sharp odor due to sulphur compounds	Thin, mobile to very viscous	0.8350 to 0.9655		
Asphaltum-base residuum	Dark brown to black	Asphalt	Thick, viscous	0.9080 to 0.9689		
Stove oil	Greenish, transparent, to dark brown	Kerosene	Thin, mobile to thick, mobile	0.8246 to 0.8854		
Coal-tar creosote	Dark, reddish brown to black	Creosote or phenol-like odor	Mobile	1.0187 to 1.0456		1.00 to 35.00
Coal-tar oil	Dark, reddish brown to black	Penetrating creosote odor	Very mobile to sluggish	0.9695 to 1.0985		1.5 to 24.0
Crude coal-tar	Black	Coal-tar odor	Fairly mobile to thick viscous	1.1363 to 1.2147		
Pine oil	Light straw yellow to deep reddish brown	Mild to penetrating, woody, pitchy	Thin, mobile to syrupy	0.923 to 0.941	1.4780 to 1.4925	
Wood creosote	Dark cherry red to black	Mild turpentine to sharp penetrating croesote odor	Thin, mobile to thick syrupy	0.9825 to 1.0783	1.5370*	
Wood tar	Black	Sharp, creosote	Thick, viscous	1.03*		
Crude wood oil	Dark cherry red to black	Sharp, penetrating wood-oil odor	Syrupy and sticky to thick, viscous	0.9195 to 1.0598	1.4910 to 1.5280	

* One determination only.

CHAPTER IV

MILL DATA

Processes

Introduction. Translation of laboratory results into terms of mill-scale operations is, in the usual case, less difficult in flotation than in gravity concentration, and in all cases more certain than where chemical reactions such as occur in leaching and precipitation operations are concerned. Any flotation result that can be obtained in a laboratory machine can be obtained in mill operation, if the essential laboratory conditions are duplicated. The converse of this statement is also true, except for the fact that the mill-sized machine is capable of handling a somewhat coarser feed than can be handled in the laboratory machine. Considering the essential elements of pulp treatment in detail, the translation from laboratory results to mill results will be as follows.

Average size of feed may be slightly coarser in the mill than in the laboratory or, if the grinding in the mill is carried to the same extent as in the laboratory, a somewhat better result, other conditions being equal, may be expected in the mill than in the laboratory.

Water may make a considerable difference between laboratory results and mill results and this difference may be either in favor of or to the detriment of the mill. The former will ordinarily be the case if a portion of the mill water is reclaimed and re-used. Under these circumstances it will ordinarily be found that the flotation agent brought back by the mill water will lessen, to a considerable extent, the amount of new flotation agent that it is necessary to add, and that froth will be more easily obtained with this reclaimed water mixed in. If, however, there is any considerable amount of soluble salts in the ore, or if the settling ponds are of considerable area and in an arid region and there is

any considerable amount of dissolved solids in the new water, then the salts in the water may have a harmful effect on flotation.

Flotation agents in the mill will be the same as in the laboratory except that it will generally be possible in the mill to lessen, to some extent, the proportion of so-called frothing oil in the mixture.

The peripheral speed of the agitators in agitation-type machines may, in general, be somewhat less in the mill than in the laboratory.

The air consumption per cubic foot of pulp treated in pneumatic machines will usually be less in the mill than in the laboratory. The pressure on the under side of the blanket will be necessarily higher in the mill machine than in the laboratory machines described, on account of the greater head on the pulp side of the blanket.

Time of treatment necessary in the mill will be very closely the same for a given recovery and grade of concentrate as in the laboratory.

The grade of final concentrate obtained in the mill will be close to that obtained in the laboratory. The recovery will come close to the indicated extraction calculated by the formula (page 166) from laboratory results, if, in the calculation, the figure for grade of concentrate is that obtained from the cleaner operation, the figure for rougher tailing is that obtained from the rougher operation, and the middling or cleaner tailing obtained in the laboratory is disregarded, provided that the grade of this middling product is not more than twice the grade of the original heads, and that the mineralogical character of the middling is not markedly different from that of the original feed.

Mill Tests. It cannot be too strongly urged that before a mill is erected, some testing work be done on mill-sized flotation machinery. This work should be done in a test mill at the mine, on ore whose prior handling corresponds as closely as possible with the scheme to be followed in the finished mill, and the water used should be as near as possible of the character of the water that is to be used in the operating plant. If such a test does no more than confirm the laboratory results, it will

pay for itself in the information that it gives concerning mill operation on the ore and it may be that the test will bring up conditions which were overlooked in the laboratory testing work. Some of the equipment used in such a test can ordinarily be utilized in the final plant so that it need not all be charged against the testing work.

Equipment and Processes

Skin Flotation Machines

The Wood machine, as described in U. S. patent 1,088,050, is illustrated diagrammatically in Fig. 24. It consists of two tanks (*A*) and (*P*), in which water is maintained at the desired level by means of regulating valves. A roller (*C*) covered with corrugated rubber belting and submerged with its center well below the surface of water in the tank (*A*) is caused to rotate in the direction indicated by the arrow. As the roller emerges from the body of liquid, it carries with it, covering its upper surface, a thin layer of water. Dry ore from the hopper (*H*) is fed in a thin sheet by means of the shaking feeder (*G*) onto the surface of the revolving roller. As this sheet of dry ore strikes the surface film on the layer of water on the roller (*C*), the gangue minerals are wetted and sink beneath the surface of the water and settle in the grooves, while the minerals of metallic luster tend to float. When the floating and submerged minerals are carried over to the point where the surface of liquid in the tank intersects the surface of the roller, the floating mineral rides out onto this surface because of the fact that the surface film is continuous over the tank and the roller, while the gangue minerals, being already submerged, remain beneath the surface. As this portion of the roller passes down to the lowest point in its revolution, the submerged gangue minerals fall off and settle to the bottom of the tank. At the side of the tank (*A*) opposite from the roller (*C*) is another roller (*I*) supported with its axis above the surface of the liquid in the tank. Over this roller passes an endless rubber belt (*K*) which passes in turn over the pulley (*R*), similarly submerged in the tank (*P*), and the guide roller (*M*). The purpose of this combination of pulleys and

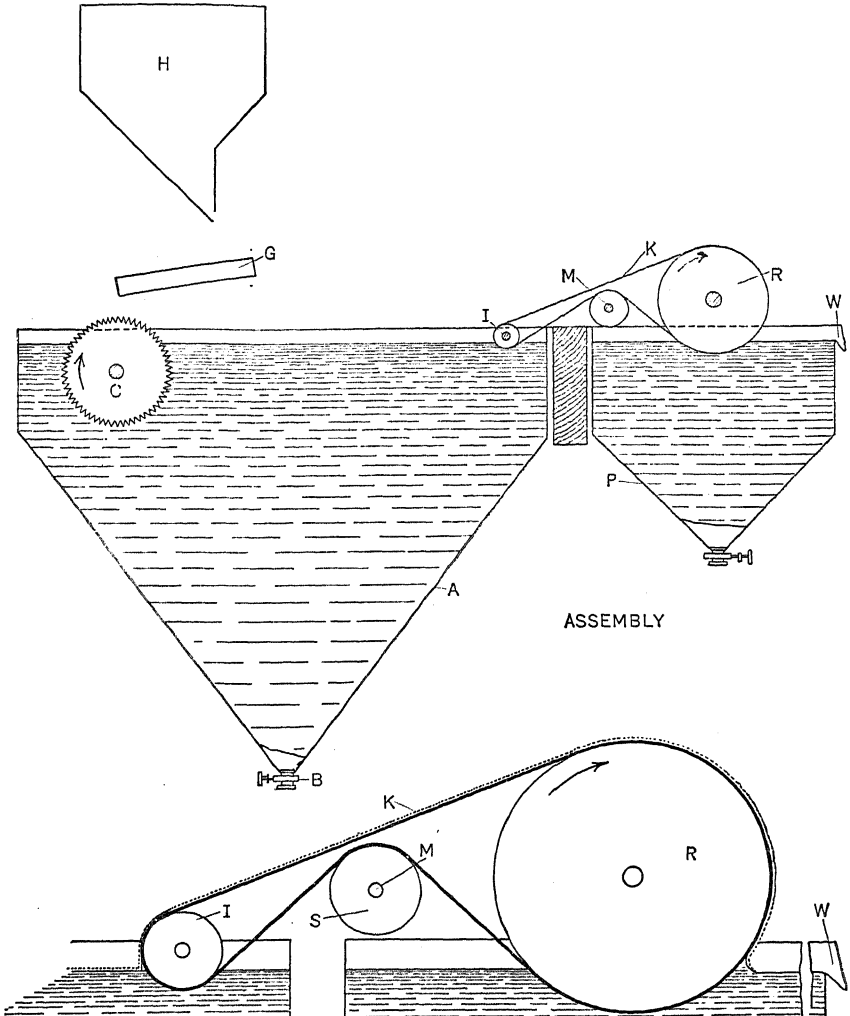


FIG. 24
Wood skin-flotation machine

belt is to remove the surface film with its load of metallic mineral from the tank (A) and to transfer the same to the tank (P). This is accomplished as shown in the large-scale sketch of this part of the apparatus in the figure. A gentle current from (C)

toward (I) is maintained by reason of the constant addition to the surface film at (C) and a constant removal of surface film at (I) by the traveling belt (K). It will be noted that the surface film is again continuous from the liquid in the tank (A) to the surface of the liquid in the tank (P) over the surface of the belt (K). Due to the disturbance at the point where the belt (K) passes below the surface of the liquid in the tank (P) the less tightly held material in the film is shaken out and settles to the bottom of the tank (P). This material constitutes the middling of the process and is re-treated on gravity concentration apparatus. The tailing of the process is discharged at the valve (B). The floating concentrate overflows from the tank (P) at the lip (W), the level of liquid in tank (P) being maintained so as to overflow a thin sheet of liquid at this point. In an earlier patent, 984,633, issued in 1911, Wood states that in certain instances a small quantity of oil "will render the film more characteristically selective with regard to the particular particles which it will convey upon its moving surface, and thus extend the range, and permit of better control of its selectivity." The machine as built has a feed roller three feet wide, requires about 0.25 h.p. to operate and is said * to have a capacity of from 1000 to 2000 lb. per hour, the higher figure corresponding to an ore with which the ratio of concentration is high.

The Macquisten tube is described in U. S. patents No's 865,194, 865,195 and 865,260 to A. P. S. Macquisten. A diagrammatic sketch of the apparatus is presented in Fig. 25. It consists of a tube (b), one foot internal diameter by six feet long, closed at the inlet end with the exception of a small central opening, and open at the discharge end. The tube is supported horizontally by means of tires on rollers (c) and is caused to rotate at about 30 r.p.m. At the discharge end a water-tight joint is made with a pointed box (d), which is fitted with an overflow lip (e). The interior of the tube is fitted with a helix of about $1\frac{1}{2}$-in. pitch. Various forms of inside surface are represented in the tube sections shown in the figure. The overflow lip of the discharge box is at an elevation of about three inches above the bottom

* "The Wood Flotation Process," H. E. Wood, Trans. A. I. M. E., 1912.

of the tube and the pulp level in the box and tube are maintained at such a level that a film of water about $\frac{1}{32}$ in. deep overflows the discharge lip. Liquid pulp is introduced into the machine

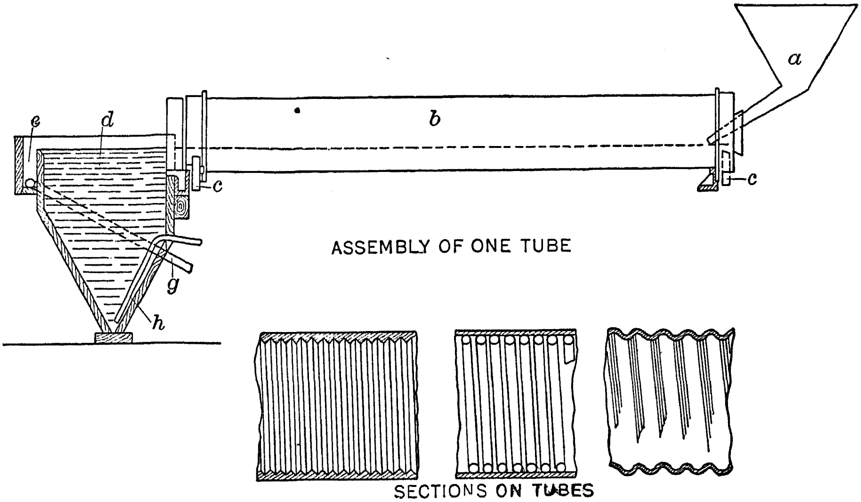


FIG. 25
Macquisten skin-flotation machine

through the feed trough (a). Due to the revolution of the tube, the solids in the pulp are raised above the surface of the pulp in the tube. Water drains away most rapidly and completely from the particles of metallic luster. When the solid material lifted above the pulp surface has attained such a height that the angle of repose is exceeded, it slides back. The dried minerals of metallic luster tend to, and do, in part, float, while the wet gangue minerals submerge. This operation is repeated many times during the passage of the pulp through the machine for every particle of solid that settles at a sufficient rate to bring it in contact with the inner surface of the tube or with the mass of settled solids thereon. The finest material or slime tends, in large part, to pass through the tube in suspension and thus to get no chance to separate. A gentle current (about 10 ft. per min.) of the surface is maintained through the tube from feed to discharge end by reason of the incoming stream of pulp. The

movement of the surface layer may be accelerated by air jets directed toward the discharge end. When the submerged solids reach the discharge tank they sink to the bottom and are withdrawn as tailing, while the floating concentrate passes over the lip of the tank into the concentrate launder and into collecting tanks through a pipe (*g*). The pipe (*h*) serves for the removal of tailing for re-treatment in another tube of the same variety. The capacity of a tube is said to be about five tons per 24 hr., but the capacity actually is determined more by the liquid surface available for flotation and the amount of floatable material in the feed than by the quantity of solids that the tube will convey. At the Morning mill * at Wallace, Idaho, 175 to 200 lb. of zinc concentrate can be floated per tube per 24 hr. At this mill feed pulp is passed through four tubes in series, three tons per 24 hr. being passed to each series. This makes the average capacity per tube 0.75 ton per 24 hr. The feed is granular having about 9 per cent. on 40-mesh and 11 per cent. through 200-mesh. The feed pulp carries from 14 to 20 per cent. solids.

Macquisten states in Patent 865,194 as follows: "Usually water would be employed as the separating agent in the case of metalliferous ores, but obviously any liquid may be substituted therefor, which has the suitable constitution or properties to effect the separation in the manner herein described, or the properties of the water or other liquid with respect to its surface tension or capillarity may be modified by the addition of a suitable acid, or alkali or soluble salt or other substance. The surface condition of the particles to be separated may be modified or altered by suitable treatment with active chemicals which will attack the surface of the particles." In practice, petroleum oils and acids have been premixed with the pulp before flotation is attempted.

The DeBavay process of skin flotation is described in U. S. patent 864,597 issued to A. J. F. DeBavay. A process called the DeBavay and practiced on a considerable scale in Australia, is described by Hoover ("Concentrating Ores by Flotation,"

* O. B. Hofstrand, Bul. A. I. M. E., May, 1912.

3rd edition, page 114). This method bears little, if any, resemblance to the method described in the patent. The patented method is based upon a conception of DeBavay's, stated by him in his patent as follows: "I have found that the sulfids of zinc, lead and silver as they exist in the ore are generally coated with carbonates of zinc, iron and manganese and other matter and that it is impossible to separate by flotation in such a condition more than a very small proportion of the zinc blende particles from the gangue and even then the floatable portion contains a large proportion of the gangue, and hence it becomes necessary to treat the ore as herein described to prepare it for separation by flotation." The method which he describes consists in subjecting the pulp to the action of an equal volume of a weak aqueous solution of carbonates of ammonia, bi-carbonates of sodium or bi-carbonates of potassium or of carbonic acid gas, or of any other reagent which will bring about the separation of the coating or covering from the zinc blende particles, or to trituration, and subsequently delivering the ore, after washing out this solution and the slimes, as a thin paste upon the upper end of an inclined table and washing down this table with a thin sheet of water onto the surface of water in a tank. In this tank the sulphide minerals were supposed to float by skin flotation and the tailings to sink. He further specified that by the term "water" he included any other liquid in which particles of zinc blende are capable of flotation. In the process described by Hoover the tailing from gravity concentration, crushed to pass about 40-mesh, was first de-slimed, then fed into a mixing tank and agitated with a cold acid solution of about 0.2 strength, the proportion of acid solution to solid being four or five to one. After treatment for a considerable time in this tank the solid was allowed to settle, the acid solution drawn off and the settled solid was washed twice to remove acid. The solid was next placed in an "oiling vat" in which it was thoroughly agitated with water and from two to three pounds per ton of a mixture of one part of castor oil and four parts of low-grade kerosene. A small amount of chlorine gas (about 0.02 per cent. on the water) was also passed into the mixture in this tank. The oiled pulp was

elevated from this machine by means of a montejus and fed into a series of separating cones of the variety shown in Fig. 26. The

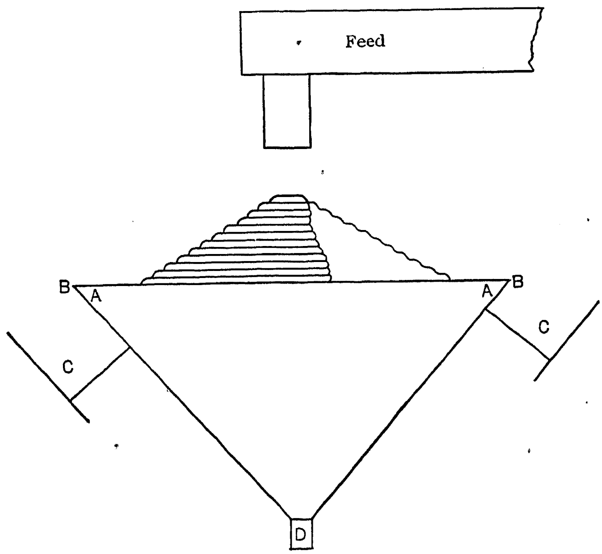


FIG. 26
DeBavay separating cone

pulp flowing down over the corrugated conical surface, upon meeting the liquid surface of the pulp in the separating cone at (*A*), divided into sulphide-rich material which floated and was removed over the periphery, and gangue which sank and was removed through the spigot. A large number of these cones were used, the capacity of each being small.

In carrying out this process in the laboratory it is to be observed that the floating material consists principally of agglomerates of sulphide minerals and air bubbles and that the process is rather one of pulp-body froth flotation than of skin flotation.

Oil-flotation Processes

The Everson process, described in U. S. patent 348,157, consists essentially in mixing dry, finely-powdered ore with oil and subsequently diluting with acidulated water to form a freely-flowing pulp, and agitating and separating the sulphide mineral.

: relative proportions of ore and oil in the mixing operation such as will form a pasty mass. The next step is described :he patent as follows: " . . . the mass [should] be opened or broken up and thoroughly stirred in the water in order t the sand or quartz may be freed and carried away." There further instruction to remove the concentrate by constant rflow of water or by other devices. The result of the treat- ıt as above outlined is to produce a mass consisting essen- ly of oil, sulphide mineral and air, which is lighter than the gue and can be separated therefrom partly by buoyancy ıe and partly through the aid of a rising current of water. : disclosure of the patent as to reagents is broad. In general s:—" . . . a fat or an oil, either animal, mineral, or vege- le, or a fatty constituent or acid of an animal or vegetable or oil, or any constituent of a mineral oil, together with an l, either mineral or vegetable, or a soluble neutral or acid : . . ." Further:—" . . . any fat or oil, and any acid, ıer mineral or vegetable, or any soluble neutral or acid salt, any compound of fats and oils with appropriate acids. . . ." cifically:—" . . . petroleum . . . paraffine-oils . . . tallow, elted,) lard, lard-oil, red-oil, (impure oleic acid,) cotton-seed castor-oil, sperm-oil, and linseed-oil, and some combina- ıs of these with each other. The acids . . . are sulphuric, lrochloric, nitric, phosphoric, acetic, oxalic, tannic, and gallic. . the following salts, to wit: the sulphates and chlorides sodium, zinc, and copper, and the double sulphate of potash l alumina." A method of making and using sulphonated oils lisclosed.

Che **Robson process,** as described in U. S. patent 575,669, con- :s essentially in mixing powdered ore in a moist or pasty state, containing from 25 to 35 per cent. water with a considerable k of an oily liquid, and subsequently to permit the mixture stratify, when the oil carrying the minerals of metallic luster l float above the balance of the mixture and may be drawn with its mineral load.

Che **Elmore process** described in U. S. patents 676,679 and),070 and further described as to apparatus in patents 653,340

and 692,643 consists in first producing a freely-flowing pulp by mixing pulverized ore and water in proportions of 6 to 1 to 10 to 1 by weight, adding thereto a relatively large quantity of oil, up to more than a ton per ton of solids, adding also sulphuric acid, mixing the ingredients together in a trough in which rotates a horizontal shaft with blades, and then passing the mixture to a separating box of the nature of a spitzkasten. The pulp level in the separating box should be kept at such a height that slight overflow of pulp liquor is allowed. Under these conditions the oil layer on the surface of the pulp will be not in excess of one half inch in thickness. Elmore specifies that the mixing should be so limited in violence as not to break the oil up into minute globules, i.e. not to emulsify the oil in the pulp. In the mill applications of the Elmore process it has been found that a considerably greater quantity of sulphide is floated than can be accounted for by the buoyant force that could, under the most favorable circumstances, be applied due to the difference in specific gravity between the active oil and the pulp, and examination of a floating concentrate reveals the fact that a considerable amount of air in the form of minute bubbles is present. In practice, the actual oil consumption per ton of ore treated is relatively small and is said to be between 10 and 20 lb. per ton of ore.

Scammell and Wolf, in U. S. patents 770,659 and 787,814 respectively, describe methods of oil flotation in the presence of a freely flowing pulp in which the viscosity of the oil is increased by various methods. The principal method described is treatment of the oil with chloride of sulphur before introduction into the pulp.

None of these oil-flotation methods is in present use.

Froth Flotation

Pulp-body-concentration Processes

The Potter and Delprat processes, now combined and known by the name Potter-Delprat process, are described in U. S. patents 735,071, 763,662, 768,035 and 776,145. As at present practiced the dewatered pulp is fed into an apparatus of the type shown

in Fig. 27, which is from 10 to 20 ft. deep. Hot sulphuric acid solution of 2 to 3 per cent. strength or acid salt cake solution of 1.3 to 1.4 density is introduced through the pipes shown to

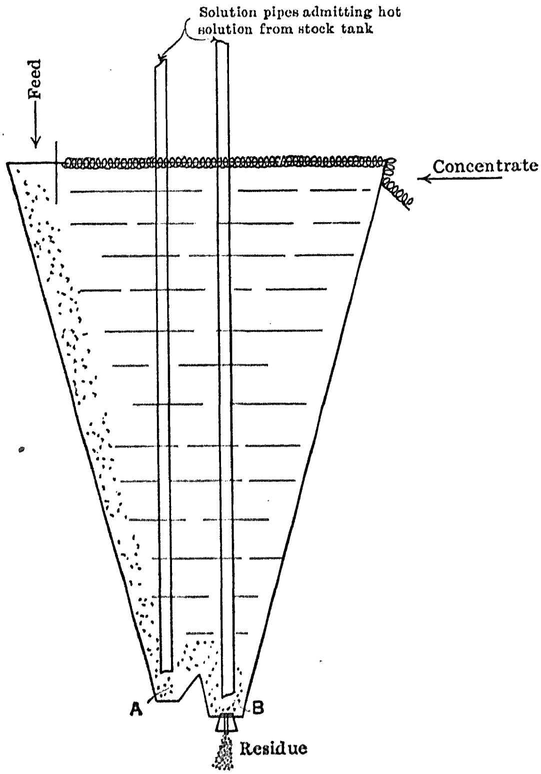


FIG. 27
Delprat frothing box

the bottom of the vat. A layer of solids two feet or more in thickness, depending upon the depth of the vat, is maintained in teeter above the spigot. Gas is precipitated as bubbles onto the sulphides and the bubbles rise to the surface with a load of sulphide, forming there a coherent froth which overflows as shown. The tailing is drawn off as a thickened product from the spigot at the bottom of the box. The compartment without a spigot is for the purpose of collecting any coarse particles which would tend to clog the spigot. This process has been

used for years in the treatment of large tonnages of sphalerite ores in Australia and has made good recoveries in the form of high-grade concentrates. In ores containing carbonates, as do most of the Broken Hill ores where these processes were invented and practiced, it is probable that most of the effective gas is carbon dioxide. Air and water vapor will, however, precipitate in sufficient quantity, under the conditions of the practice of the combined processes, to effectively raise the sulphide in the form of a coherent froth.

The Froment process was discovered in 1902 and patented in Italy and England but not in the United States. The steps of the process, as described by the inventor, are essentially as follows: two and one half parts of ore with six parts of water and from 1 to 2 per cent. of limestone are introduced into a mixer with an amount of oil ranging from 1 to 1½ per cent. on the ore and stirred thoroughly to disperse the oil through the pulp. The mixture is then run into a second vat fitted with a slow moving rake at the bottom and sulphuric acid sufficient to react with the limestone is slowly added through a perforated coil. The sulphides rise to the surface in the form of a heavy matted froth and may be skimmed off. As in all other flotation processes the ore must be finely ground.

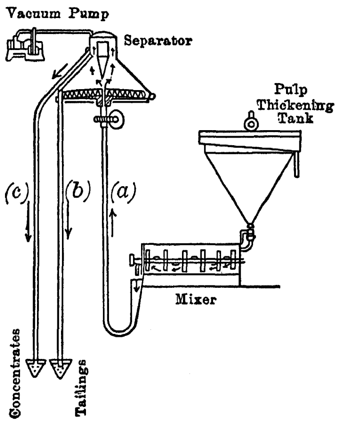


FIG. 28
Elmore vacuum plant

The Elmore vacuum process, as described in U. S. patent 826,411, is practiced in a plant such as is shown in Fig. 28. Pulp ground to at least 0.5-mm. maximum size and containing about 50 per cent. solids is mixed with oil in an amount less than 0.5 per cent. on the ore, with or without a correspondingly small amount of acid, in a mixer and from thence discharged into the feed pipe (*a*) of the separating apparatus. Here water is added to bring the pulp to a consistency of 15 to 25 per cent. solids.

The separating apparatus consists of a closed conical chamber fitted with a slowly revolving rake at the bottom, a tailing discharge pipe (*b*) at the periphery and a concentrate discharge pipe (*c*) from near the apex. The separating chamber is attached to a vacuum pump. The lower end of the pipes (*b*) and (*c*) is sealed by causing them to discharge below the surface of liquid in tanks as shown. The vertical lift in the pipe (*a*) is about 25 ft. The vertical length of the pipes (*b*) and (*c*) is somewhat over 30 ft. A vacuum of 25 to 27 in. of mercury is maintained. Under the influence of this vacuum the pulp fed into the pipe (*a*) passes up into the separating chamber. Here air comes out of solution at the surfaces of the sulphides and raises them through the liquid in the separator to the apex of the same where they overflow into an annular launder and pass down the pipe (*c*). At the same time the tailing is slowly scraped by the rakes to the periphery of the floor where it passes down the tailing-discharge pipe (*b*). The rate of flow in pipes (*a*) and (*b*) is so regulated that the pulp level is maintained slightly below the overflow lip in the apex of the separator. The capacity of a 5-ft. separator is from 25 to 50 tons of ore per day. No retreatment of the concentrate is necessary. The power consumption per pan is small, being well under 5 h.p. for mixer and vacuum pump together. The development of the vacuum process was stopped by the introduction of the agitation-froth process but it is probable that if the same amount of work had been expended in attempts to make the vacuum process a highly efficient operation, as was spent on bringing the agitation-froth process to its present degree of efficiency, the result would have been equally favorable.

Agitation-froth flotation is practiced in several varieties of machines. The most widely used are the Janney mechanical and the Minerals Separation.

The Janney machine is shown in Fig. 29. It consists essentially of an agitating compartment (*a*) with two froth-separating compartments (*b*). In the usual and best form the agitator shaft is mounted as an extension of the spindle of a 10-h.p. vertical motor as shown. The agitator shaft carries two four-armed impellers with blades set at 45 degrees. The agitating

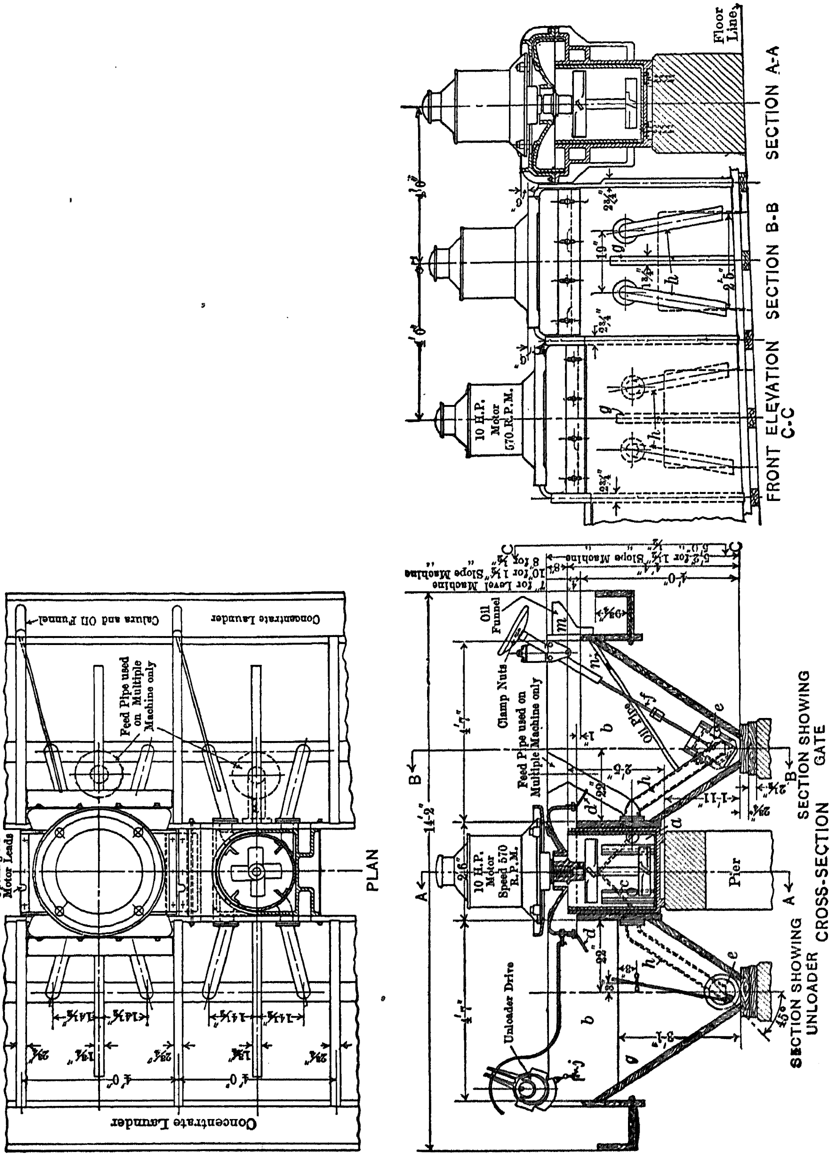
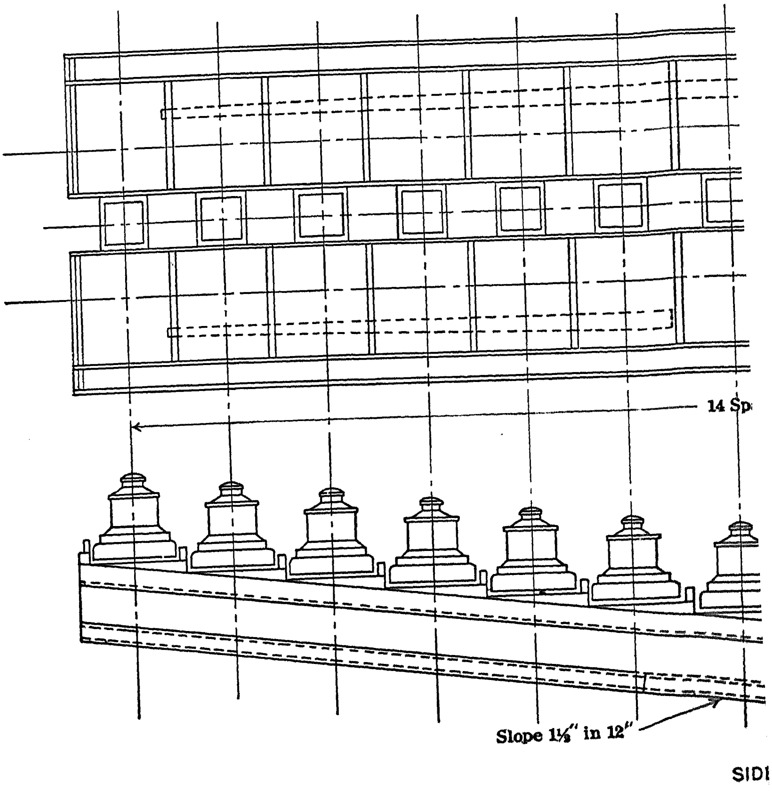
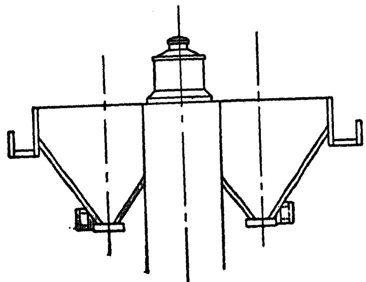


FIG. 29
24-in. Janney mechanical flotation machine

compartment is circular and contains four baffles (c) extending slightly more than one-half the distance from the bottom toward the top. The arms of the lower impeller are shorter than those





Fifteen-compartment Janney mech

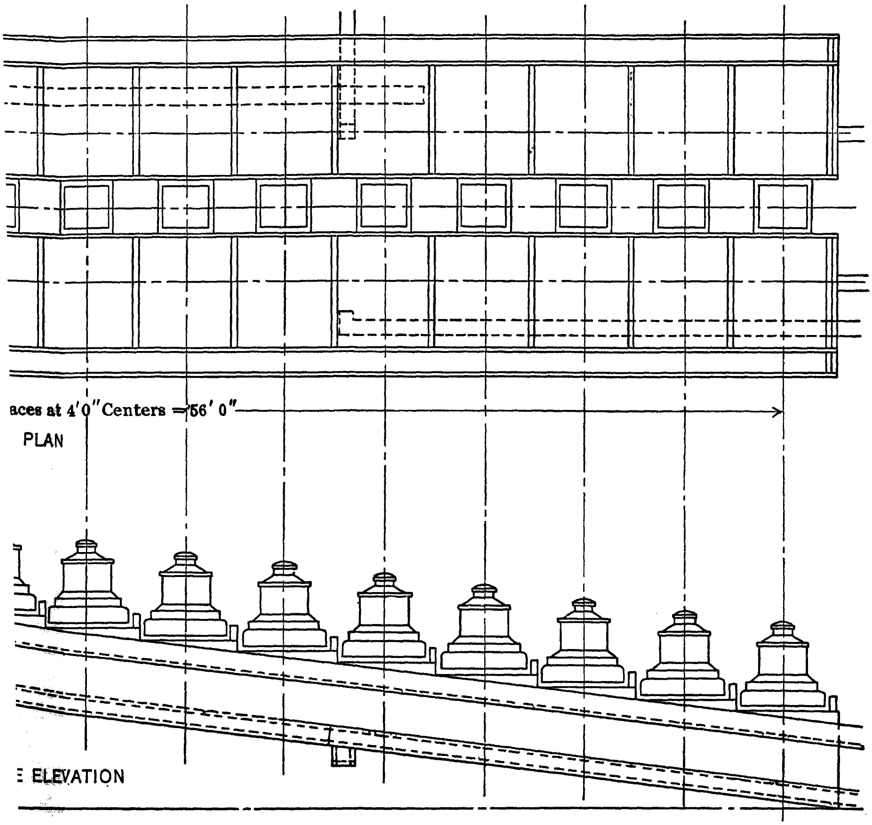


Fig. 30
anical agitation machine, series arrangement

of the upper in order to clear these baffles. Feed is introduced through the side of the compartment, near the bottom, by means of a pipe; is thrown out through the channels at the top, on each side of the agitator compartment, and is introduced, by means of the submersion blades (*d*) slightly below the level of the pulp in the froth-separating compartments. In general, several machines are installed in series. Outline arrangements of two such installations are shown in Figs. 30 and 31. In the arrangement shown in Fig. 29 the tailing leaving the first compartment, passes through an opening (*e*) regulated by means of the valve (*f*) into the froth-separating compartment in the succeeding machine. The froth-separating compartments are divided to within about a foot of the overflow lip by means of a wall (*g*). From the bottom of the two compartments thus formed, pipes (*h*) lead back to the agitating compartment. Pulp, entering the first or upper of these two sub-divisions in a given froth-separating compartment, is drawn upward into the agitating compartment, thrown over the top of the same into the froth-separating compartment and falls back, a part on each side of the divider (*g*). Practically all of that which falls back on the upper side of the divider is again drawn up through the pipe from that compartment into the agitating compartment. A part of that which falls on the down-stream side of the divider (*g*) is also drawn back into the agitating compartment. Thus a part of the pulp is circulated in each machine and subjected to agitation and aëration more times than would be the case if the flow were alternately through agitating and froth-separating compartments. Froth is skimmed by means of the eccentrically driven unloader (*j*) which is operated by means of a small independent motor (*k*), Fig. 30. The series of machines is ordinarily preceded by one, two, or three agitating compartments built without froth-separating compartments. These are called "emulsifiers." Flotation agents are added to the pulp entering the emulsifiers. Additional flotation agents are added, if necessary, through the oil funnel (*m*) and pass through the oil pipe (*n*) into circulating pipes (*h*). The multiple arrangement of feed shown in Fig. 31 is ordinarily used where it is desired to make

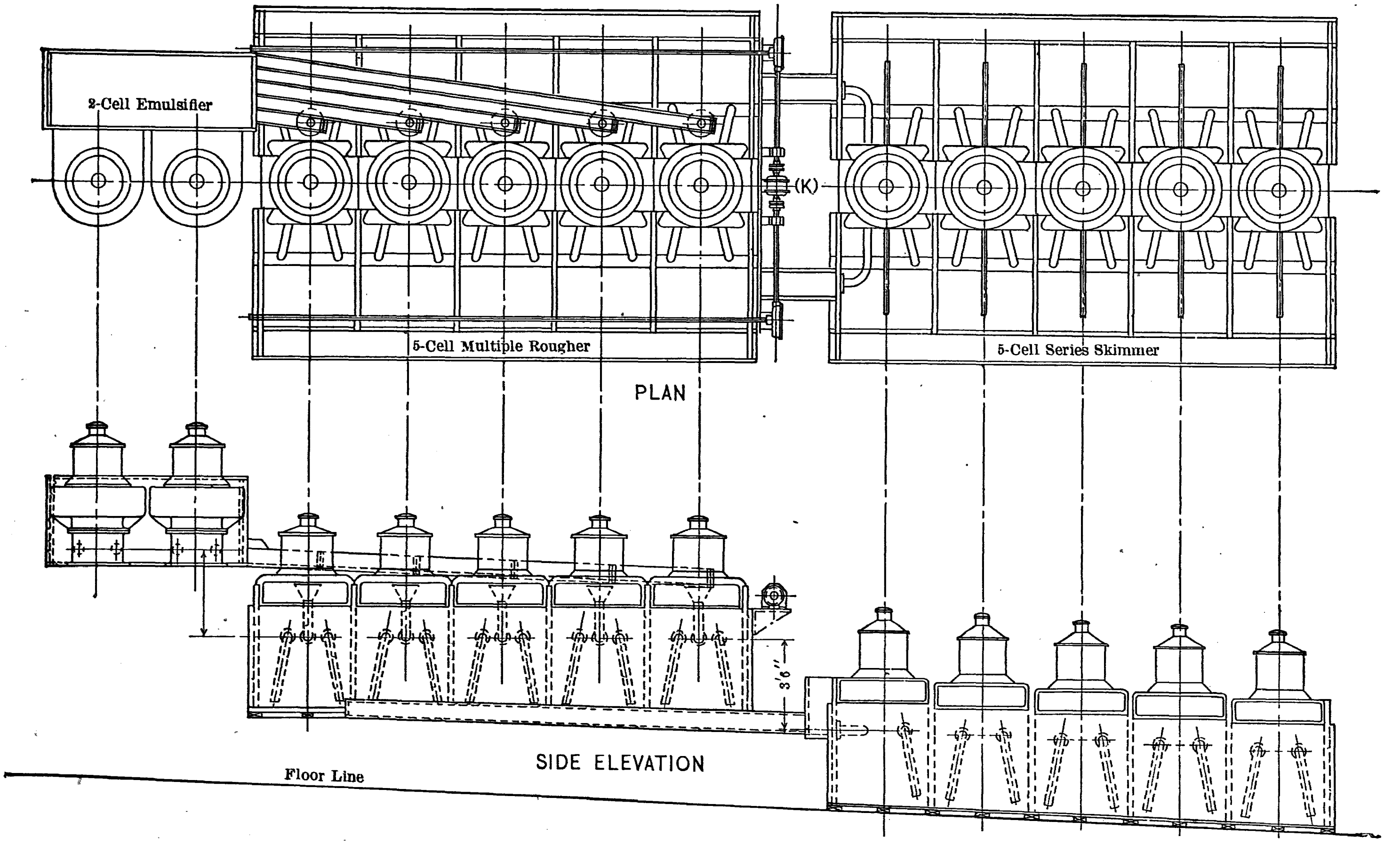


FIG. 31
Ten-compartment Janney mechanical agitation machine, multiple-series arrangement.

a finished concentrate on the early cells and to circulate the froth from the later cells back to the head of the series. The series arrangement shown in Fig. 30 is used where the froth taken from the various froth-separating compartments is cleaned on other machines.

From five to fifteen agitators, each with two froth-separating boxes and preceded by one or two emulsifiers, comprise a unit. The size of a unit is indicated by the diameter of the agitation chamber and the number of single machines in series. A 24-inch, 16-compartment unit consists of one or two 24-inch emulsifiers followed by 16 agitating compartments with cross-armed impellers, the upper 20 inches and the lower 14½ inches tip to tip, respectively, with 32 froth-separating boxes. The capacity of such a machine depends upon the percentage of solids in the pulp, the number and diameter of impellers, and the character of the ore. On a silicious sandy copper ore in pulps carrying from 10 to 28 per cent. solids, a 13-compartment, 24-inch machine has a capacity of from 150 to 550 tons per 24 hours, the relation between tonnage and percentage of solids being represented by a straight line. Decrease in the number of cells in series will mean a corresponding and almost proportionate decrease in capacity, if the same grade of concentrate and percentage of recovery are to be maintained. There will also be a decrease in capacity if a slimy feed replaces a sandy feed. This is due principally to the fact that such a feed must be treated in a pulp containing a lower percentage of solids and that consequently the bulk to be passed through the machine for a given tonnage of solid is increased. The decrease in capacity is about in proportion to such increase in bulk. The power consumption of a 24-in. machine is approximately 10 h.p. per agitator. The machine is usually run at a peripheral speed of 3600 ft. per minute.

A Minerals Separation machine is shown in Fig. 32. The machine is installed as shown in the figure with agitators and froth-separating boxes on the same level. The machine consists essentially of the agitating compartment (*a*) and froth-separating compartment (*b*). The agitator, which is of the four-armed

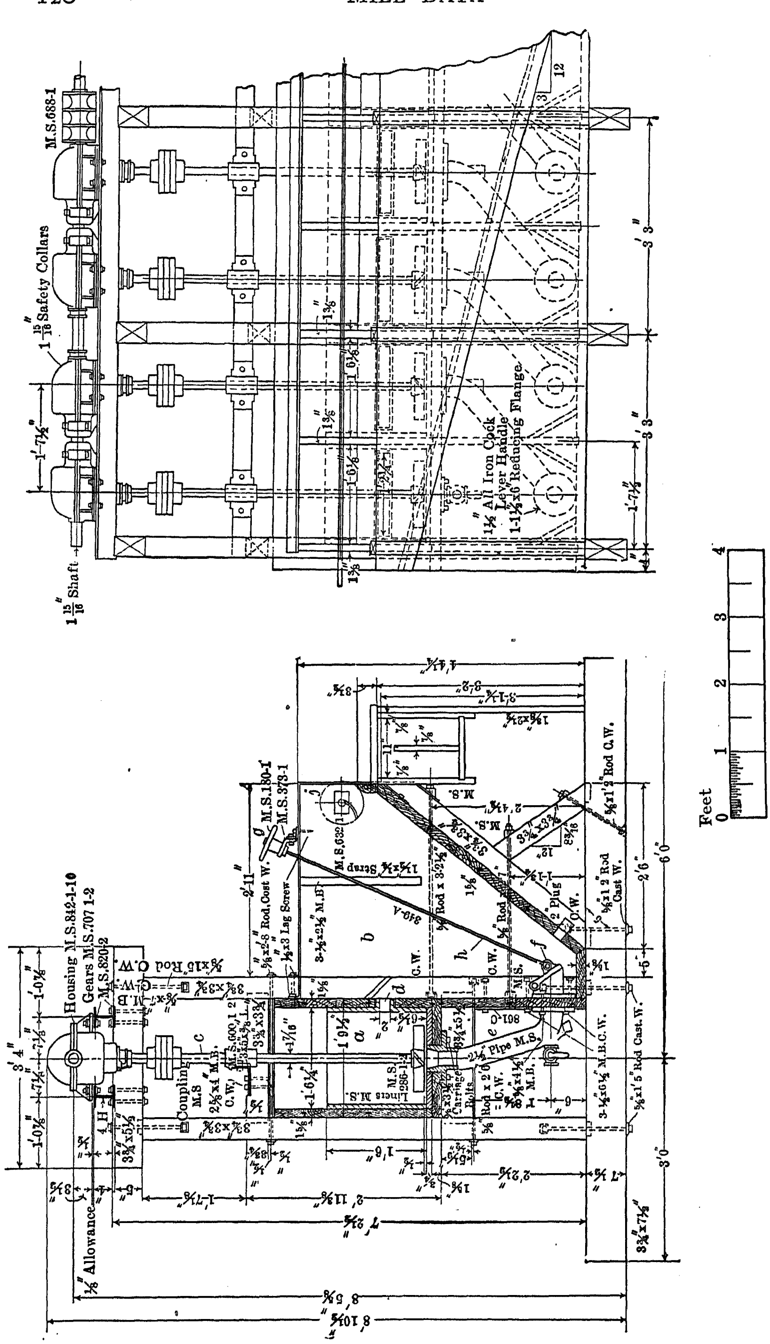


FIG. 32
Minerals Separation 12-in. standard flotation machine

cross type, with blades set at 45 degrees, is placed close to the bottom of the agitating compartment and is carried on a vertical spindle (*c*), driven through enclosed bevel gears from a horizontal line shaft. Feed pulp is introduced into the first agitating compartment, or is first passed through one or more agitating compartments without froth-separating boxes, corresponding to the emulsifiers of the Janney machine. The pulp, after agitation and aëration, is thrown out through the slot (*d*) into the froth-separating compartment, entering at a point about six inches below the pulp level. The tailing from the froth-separating compartment passes through the pipe (*e*) into the bottom of the next agitating compartment under the influence of the pumping effect of the agitator therein. The rate of flow is regulated by means of the valve (*f*) actuated by the hand wheel (*g*) and rod (*h*). Froth is removed by means of the revolving scraper (*j*). From 6 to 20 agitating compartments in series, each with one froth-separating compartment, comprise a unit. The size of a unit is indicated by the distance tip to tip of the impeller blades and the number of compartments in series. The usual impeller sizes are 12-inch, 18-inch, and 24-inch. These machines have rated capacities of 50 tons, 300 tons and 600 tons respectively per 24 hours on silicious ores in pulps containing 25 per cent. solids. Actual capacities are about 2, 12, and 25 tons per cell per 24 hours on such a pulp, with a diminution in capacity with decrease in percentage of solids about proportional to the increase in volume of pulp. The power consumption per agitator is two to three h.p., three to five h.p., and six to nine h.p., for the three sizes respectively, depending upon the speed of the impeller and the tonnage of pulp passed through. The usual peripheral speed is from 1500 to 1800 ft. per minute.

Bubble-column Machines

Machines which perform the operation of concentration in a column of bubbles above the surface of the pulp are of three general types: (1) the pneumatic type in which air is released in the pulp through a porous medium; (2) the centrifugal type in which air is drawn into the pulp by centrifugal force; and

(3) the cascade or plunging-stream type, in which the air is carried into the pulp by the action of a stream of pulp falling into a body of pulp.

Pneumatic-type bubble-column machines introduce air into flotation pulps through a porous medium, which, in all present day machines, constitutes the bottom of a box or launder containing the pulp. Usually the flotation agents are pre-mixed with the feed pulp in some other machine. The best known types are the Callow cell, which was the pioneer in so far as operating mill installations are concerned, and the Inspiration type, which was developed from the Callow cell for the purpose of handling larger tonnages with a smaller amount of floor space per ton of solid handled.

The Callow cell is shown in the usual form in Fig. 33. It consists of a rectangular box (*a*) with sloping porous bottom (*b*). The usual dimensions of the box are 8 to 9 ft. long, 2 ft. to 2 ft. 6 in. wide, about 18 in. deep at the feed end and 4 ft. deep at the tailing-discharge end. The porous bottom consists of three or four layers of medium weight canvas or palma twill, supported on a screen or grid on top of an air box. The usual cell has an air box divided into eight compartments, each with an independent connection to a header, the purpose being to allow independent regulation of the air pressure on the under side of the blanket at different points in the length of the cell, in order to balance the different hydrostatic heads at different points along the cell and prevent eddy currents due to unequal air distribution. Pulp is fed into the cell behind the baffle (*c*), froth overflows the sides, and tailing is discharged through the pipe (*d*). The rate of discharge is regulated by means of the adjustable float valve (*e*). The capacity of a single unit such as illustrated is from 35 to 80 tons per 24 hours, the lower figure corresponding to a slimy ore in a pulp containing a low percentage of solids, the higher figure corresponding to a silicious, rather sandy ore in a pulp containing in the neighborhood of 25 per cent. solids. The air consumption is decidedly variable, ranging in different plants from about 6 to about 12 cu. ft. of free air per min. per sq. ft. of porous bottom, at pres-

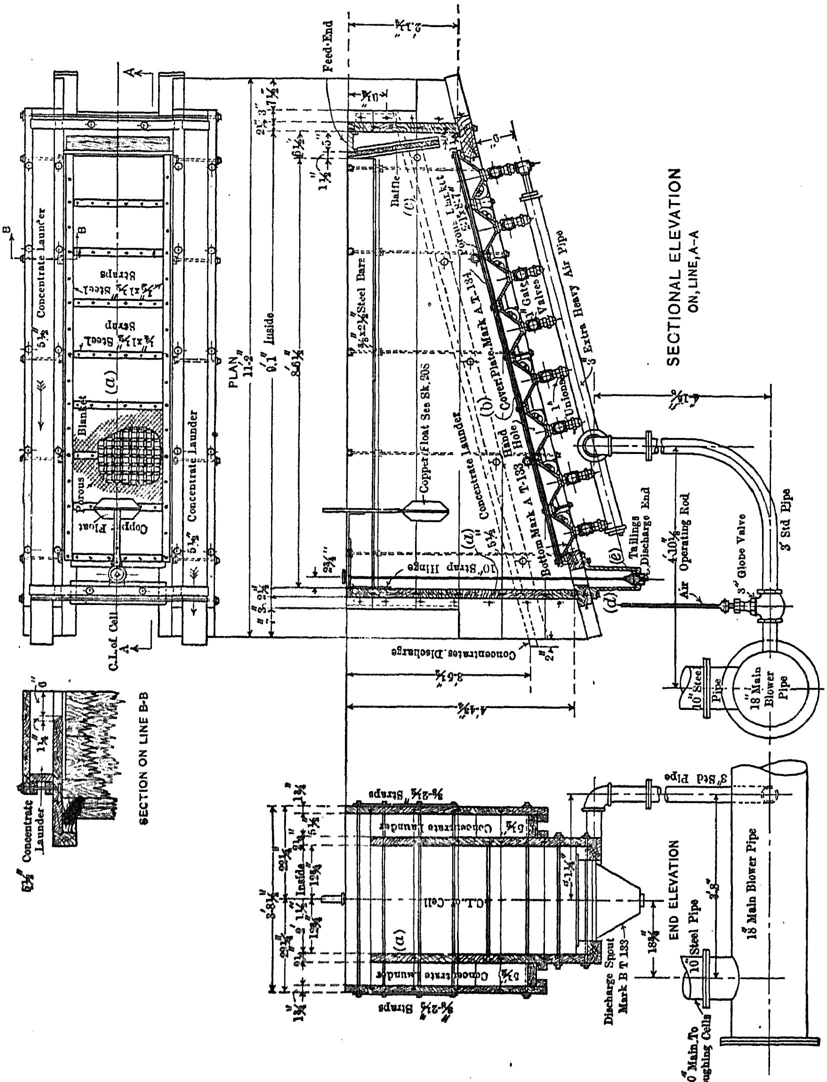


FIG. 33
Callow cell, standard mill size

sures of from three to five pounds per square inch on the supply side of the regulating valves. A good average figure is probably in the neighborhood of 9 cu. ft. per min. per sq. ft. of porous bottom. This air requirement corresponds to a power consumption of between three and one-half and four h.p. per cell.

Callow cells are usually run in parallel. General experience is to the effect that, whether run in parallel or in series, the capacity per square foot of porous bottom, in machines producing concentrate of the same grade and recovering approximately the same amount of mineral, is the same irrespective of the method of operation. A modified form of Callow cell designed to economize floor space is shown in Fig. 34. It consists of a rectangular box about 20 ft. long, 7 ft. wide and an average depth of 3 ft., set on a slope of about 2 in. per foot. The box is divided into eight compartments, 3 ft. 5 in. wide by 4 ft. 6 in. long, by means of a central longitudinal wall which extends to the bottom of the box, and three transverse walls. The first and third transverse walls extend to within 5 or 6 in. of the bottom of the box, the second or center wall is so arranged as to cause the pulp to overflow a weir in passing from the second to the third compartment on both sides of the cell. The purpose of this weir is to maintain the desired pulp level in the first two compartments on each side. Practically the partitions serve to divide the box into four cells. Air baskets consisting of shallow boxes with a porous top, of such size that they fit loosely into a compartment, are placed with the porous side up in the bottom of the compartment. The capacity of one of these machines is between 400 and 500 tons per 24 hours on a silicious ore in a pulp containing from 20 to 25 per cent. solids. Air consumption, and consequently power consumption, per square foot of porous bottom are about the same as for the smaller Callow cell.

The Inspiration machine is shown in Figs. 35 and 36. It consists essentially of a launder about 3 ft. wide and 4 ft. 6 in. deep, with a slope of about ½ in. per foot. It is provided with a removable segmented porous bottom and is divided into compartments by partitions spaced about 3 ft. center to center along its length. The usual number of compartments in a roughing machine ranges from 15 to 20. The compartments are provided with slots about 4 ft. wide and 6 to 8 in. high, cut about 3 in. above the bottom of the launder. The size of the opening between compartments is regulated by gates suspended by a threaded rod from a hand wheel and lug on timbers placed

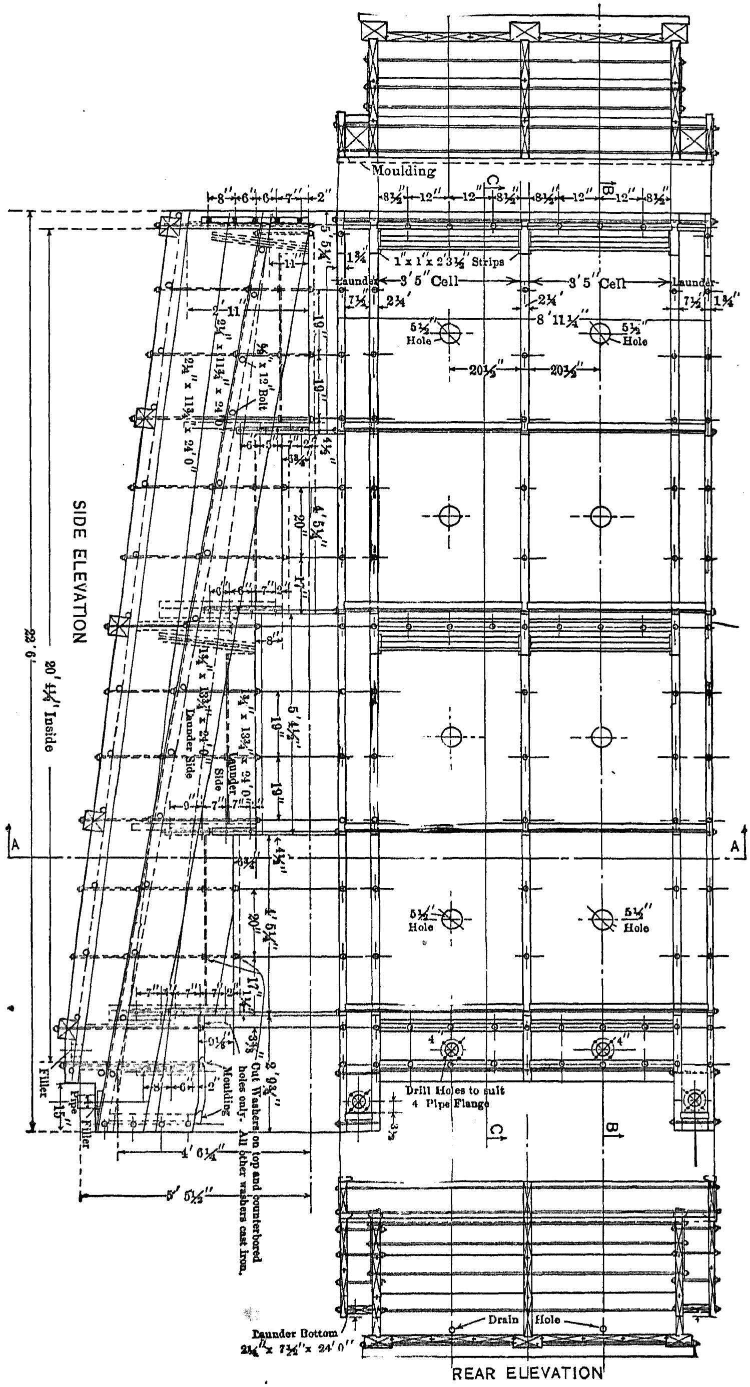


Fig. 34
Elevation of Miama type pneumatic cell

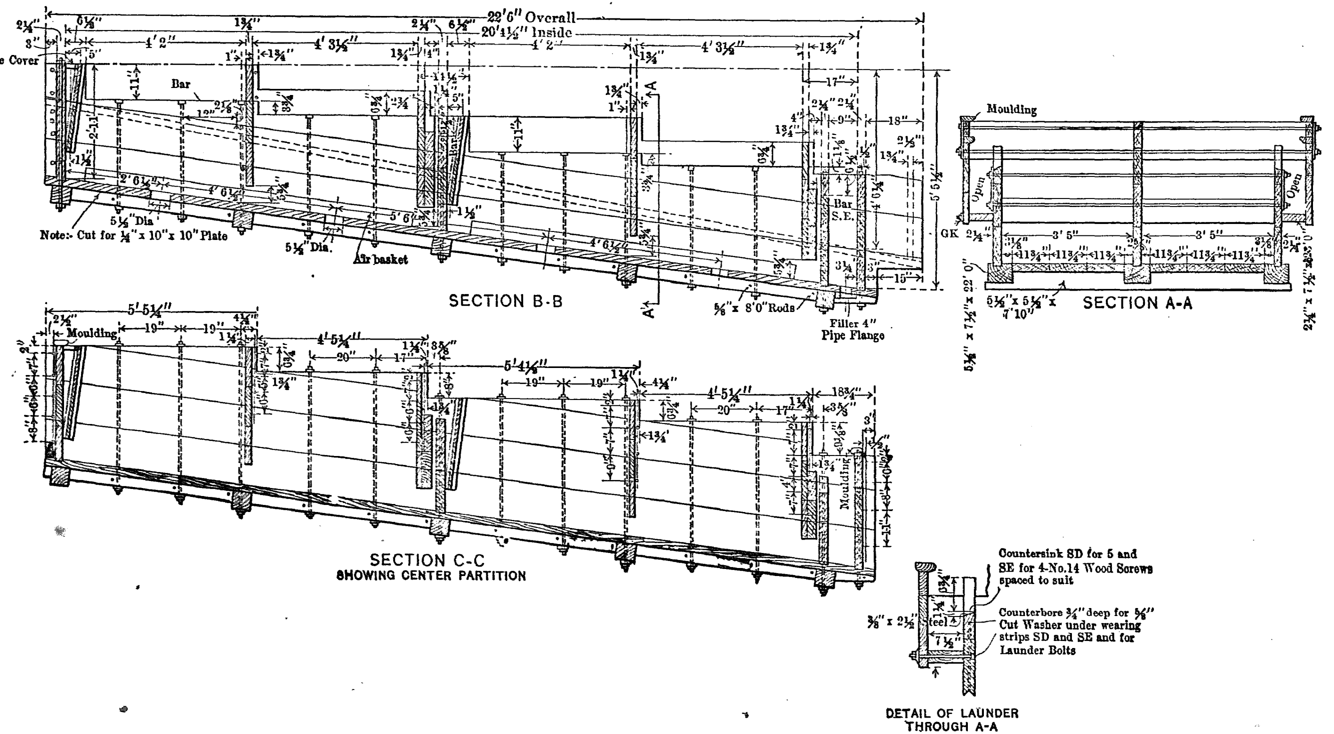


FIG. 34
Section of Miami type pneumatic cell

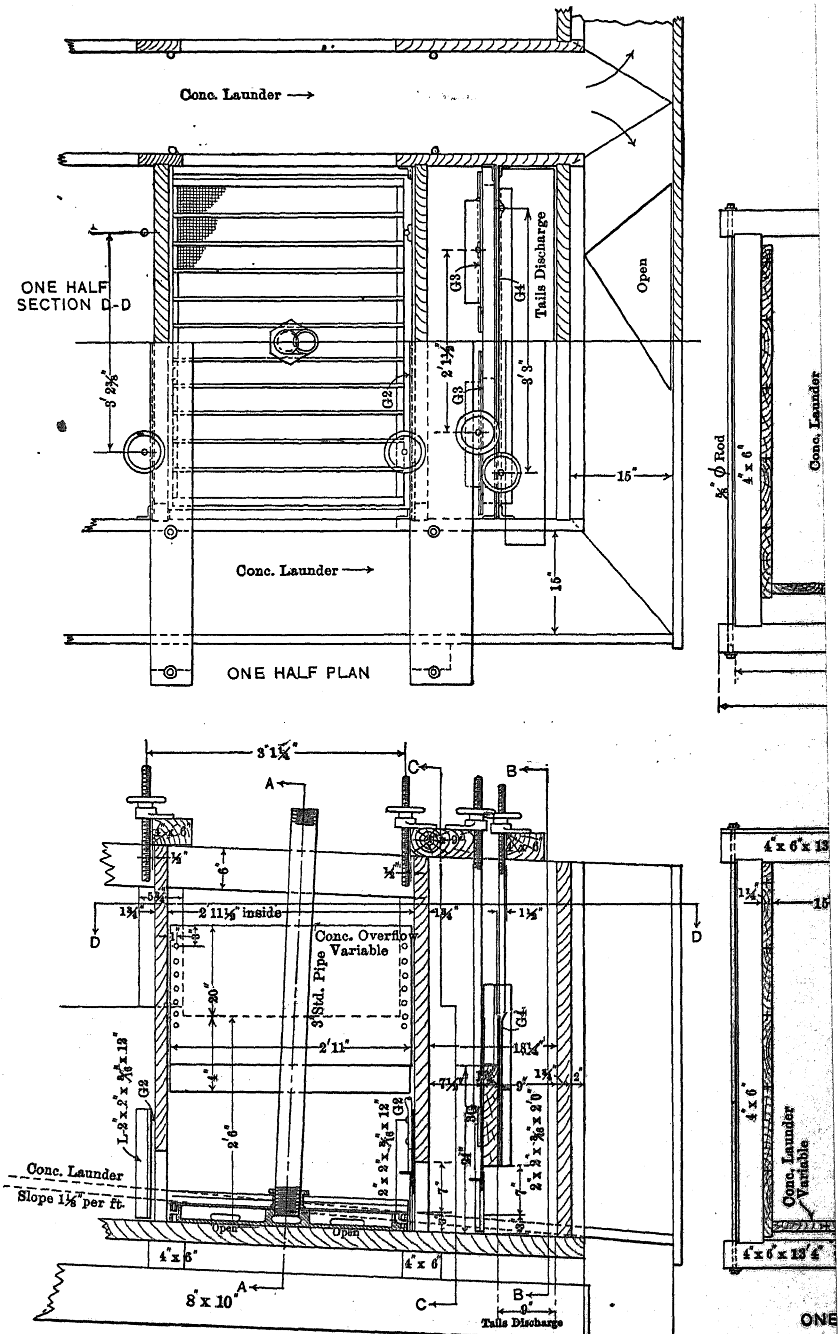
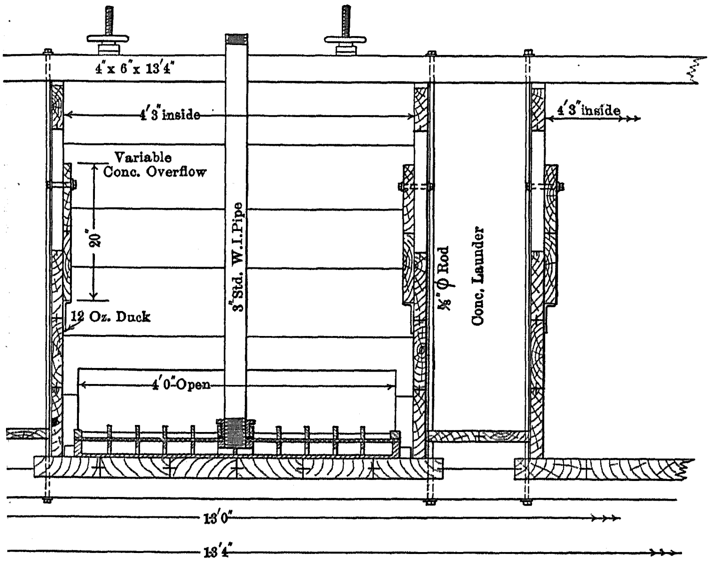
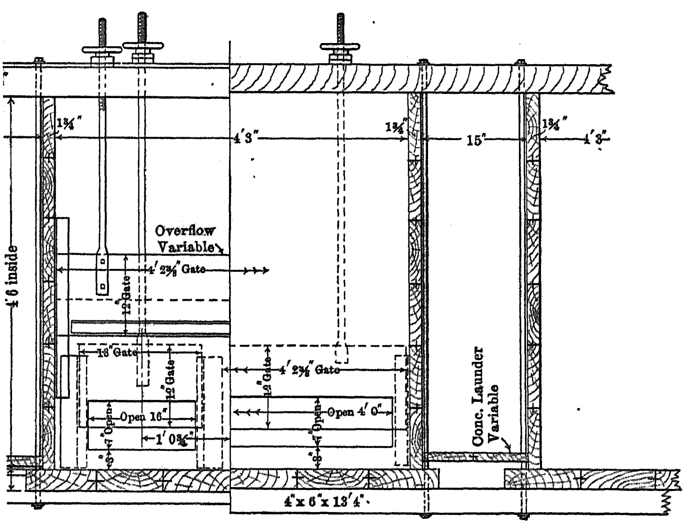


FIG. 36
Details of Inspiration



SECTION A-A



HALF SECTION B-B

ONE HALF SECTION C-C

cell

across the top of the launder. Air baskets consisting of shallow boxes of such dimensions that they make a loose fit in a compartment and with the air supply pipe coming down from the top, are placed in the bottom of the launder. In general, two such launders set side by side with a common central froth launder constitute a roughing unit. The rougher froth is cleaned in a similar smaller machine, fed by gravity from the rougher machine. The capacity of such a double unit with 16 rougher compartments each side and six cleaner compartments each side, is from 600 to 1200 tons per 24 hours, the lower figure on a slimy low-grade copper ore in a pulp containing 12 to 15 per cent. solids, the higher figure on a silicious, rather sandy low-grade copper ore in a pulp containing 20 to 25 per cent. solids. Air consumption is from 10 to 12 cu. ft. per sq. ft. of blanket surface, at from 4 to 5 lb. pressure on the supply side of the regulating valves. This means a power consumption of from 0.3 to 0.35 h.p. per sq. ft. of blanket.

The last word has not yet been said on the construction of porous bottom machines. The steep slope of the bottom in the original Callow cell and the compartmenting in the modified Callow cell and in the Inspiration cell described, were for the purpose principally of preventing sanding and consequent clogging of the porous bottoms. Machines are, however, in operation in which the box is some 20 ft. long and 6 ft. wide, with the bottom nearly horizontal, and it is reported that no serious sanding-up occurs.

In many mills brick, concrete and other materials for porous bottoms have been largely experimented with. Concrete has given satisfactory service and has been adopted in several plants. Special concrete mixtures are used and considerable skill and care in the mixing and aging are necessary, but the life of the bottoms pays for the additional expense of installation. The method of making concrete bottoms at the Ray Consolidated Copper Co. is described by H. C. McCrae in Engineering and Mining Journal, April 10, 1920, as follows:

"The cells used have inside dimensions of $35\frac{1}{2} \times 51\frac{1}{2}$ in. Two methods of installing the mats have been successfully em-

ployed by the company. The first was to construct the porous mat in place, according to the following directions:

"Build the forms in each cell of the machine so as to give a wall of ordinary dense concrete 4 in. wide and 8 in. deep. When the concrete is set, remove the forms and pour the floor of the cell with the same dense concrete to a depth of 4 in. Round pebbles varying in size from $1\frac{1}{2}$ in. to $2\frac{1}{2}$ in. are now placed in the floor of the cell to a depth of approximately 4 in., and the top is covered with small pebbles about $\frac{1}{2}$ in. in diameter to obtain a level surface. This pebble filling not only serves as a good foundation upon which to build the overlying porous mat, but also allows sufficient space to equalize the air pressure.

"The proper size of sand for use in making the air mats can be secured from the fine-crushing department of the ordinary concentrator. The sand should be as free as possible from inclosed mineral; therefore, tables treating a sand product that passes 10 mesh, with very little fines, reject tailings suitable for these mats. A hard silicious sand that contains a considerable amount of quartz is preferable. Such sand is generally of irregular fracture, and is much better suited for this purpose than the sand from soft ores or the ordinary rounded river sand. The sand is dried for screening.

"An 18-mesh copper-wire window screen placed and shaken on a horizontal plane is first used. The undersize from this screening is now passed over a screen of the same size placed at a 45-degree angle. This will remove the fines and give a clean sand of uniform size. Five parts of this sized and dry sand are thoroughly mixed with one part of dry Portland cement. This mixture is now dampened by sprinkling on water with a brush. Care should be taken only to dampen and not to add excess water, as the problem is to get the cement to adhere to the sand and dampened sufficiently to make the mass harden. Reenforcing is put in the dense concrete sides, and $\frac{1}{4}$-in. or $\frac{3}{8}$-in. iron rods, placed about 6 in. apart each way, are used in the top porous mat for re-enforcing.

"The mixture for the porous mat should now be spread over the bed of pebbles in the cell to a depth of about 3 in. A beveled

corner strip, $1\frac{1}{2} \times 2 \times \frac{1}{2}$ in., should be placed around the inclosing sides of the wall, to be removed later, and dense concrete poured in the space. This, when set, should calk any air leak along the sides of the mat. The mixture should be well tamped with a wooden mallet as filled, and surfaced with a wooden float. A metal trowel should not be employed as a surfacing tool.

"After twenty-four hours, sprinkle sparingly with water from a brush, taking care not to get too much water on any one spot. This sprinkling operation should be repeated several times each day for three or four days, then a spray from a hose may be employed, and the concrete given plenty of water. About ten days is required for the porous mat to set. Some of the machines with this porous mat made as outlined have been combined into one large cell; others have been divided into three separate and distinct compartments by means of sheets of No. 16-gage iron partitions placed in dense concrete in the floor of the air chamber and extending to the surface of the mat. The number of compartments into which each cell is divided regulates the number of valves to be used for the adjustment of air.

"In recent practice, however, it has been found more satisfactory to replace the walls and bottom of the dense concrete air chamber with a metal pan, which permits the making of the porous mats at any convenient place and reduces the time required for replacement. This metal pan is constructed in such manner as will permit of its being placed in the $35\frac{1}{2} \times 51\frac{1}{2}$-in. cells with ease. It is divided into three compartments, and each compartment into two sections to reduce the necessary size of the porous blocks. The pan is about 7 in. deep, with a 1-in. angle iron, serving as a support for the porous mat, riveted on the sides and between the sections and compartments, 3 in. from the top.

"Forms for the porous blocks are made to allow $\frac{1}{2}$-in. space between the porous mat and the sides of the pan or the angle iron, which space is to be filled with neat cement. The same specification for these air mats is followed as previously described. The re-enforcing for the porous blocks consists of two 23-in.

and three $10\frac{1}{2}$-in. iron rods, $\frac{1}{4}$-in. in diameter, and all rods are wired securely at each cross. The mixture is tamped in toe forms and allowed to set for at least seven days, then stripped and placed in the metal pans. The forms are $3 \times 10\frac{1}{4} \times 22\frac{3}{4}$ in. inside, which permits of slight extension of the re-enforcing rods into the neat cement.

"Experience with the operation of the air mats shows that about every sixty days the machine must be shut down and the mats thoroughly scraped with a small hoe, and also brushed well with a wire brush. This is necessary to remove a slight coating of iron carbonate that forms on the top surface."

Some of these bottoms have been in use for over two years.

Centrifugal-type bubble-column machines utilize centrifugal force to introduce air into the flotation pulp. The best known machines of this type are the Ruth, the Groch and the Hebbard.

The Ruth machine is shown in Fig. 37. It consists of a box (*a*) divided by partition (*b*) into an aërating compartment (*c*) and a spitzkasten (*d*). The aërating compartment (*c*) is fitted with a grid (*g*) which prevents the creation of a vortex in the upper part of the chamber (*c*). A hollow vertical shaft (*e*) open at the upper end, extends into the aërating compartment and carries at its lower end a disk (*f*) for circulating pulp and introducing air. This disk is shown in larger scale at one side of the sectional view. The revolution of the disk in the direction shown by the arrow produces vacua behind the shields over the air passages (*y*) and air passes in through the hollow shaft to fill these spaces. The rotation of the disk also causes pulp to be drawn up through the passages (*x*) from the chamber (*h*) which opens into the lower part of the spitzkasten. Separation takes place in the bubble column above the compartments (*c*) and (*d*) and concentrate in the form of froth is overflowed at the lip (*i*). In the standard machine the disk is made 14 in. diameter and the hollow shaft is 1.1 in. inside diameter. The machine is driven at from 270 to 300 r.p.m. A capacity of 150 to 200 tons per day with a power input of about one h.p. per spindle is claimed for an eight-cell machine. It is also claimed that 20-mesh material can be treated. The writer is inclined to

believe that all of these claims are decidedly optimistic and that a considerably lower tonnage of pulp ground to the usual flotation size (65-mesh) must be handled in order to get good results.

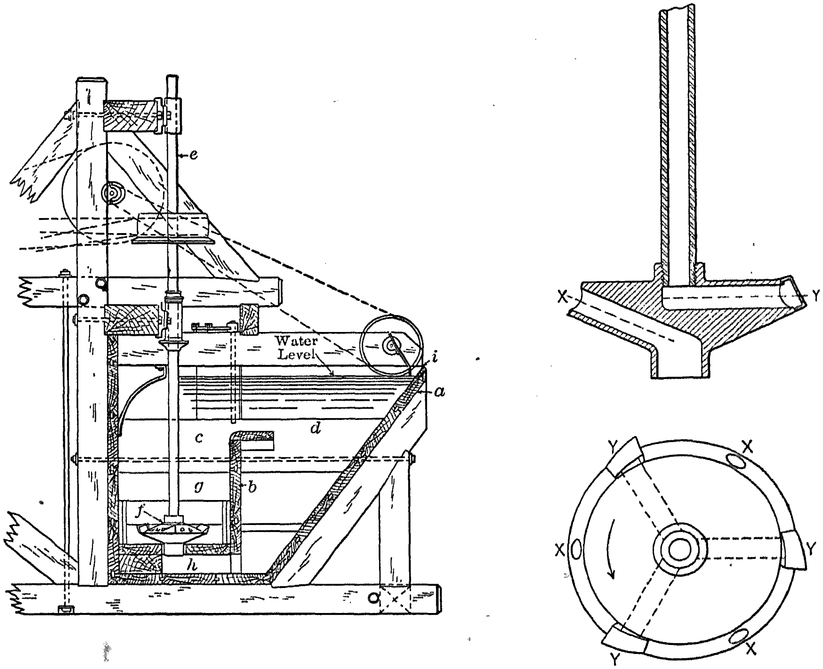


FIG. 37
Ruth flotation machine

The Groch machine is the same in principle as the Ruth machine and differs from it only in the method of construction of the rotating disk and in the arrangement of the aërating compartment and spitzkasten.

The Hebbard sub-aëration machine is shown in Fig. 38. It consists essentially of a trough (*A*) partially sub-divided into compartments by partitions (*c′*) and (*t*), in which compartments are rotated vertical spindles (*a*) carrying at their lower end disks (*b*) with radial arms on the lower face. The machine pictured is known as a 24-in. machine, so denominated by the diameter of the disks. The trough (*A*) is 3 ft. wide, 24 ft. long, and

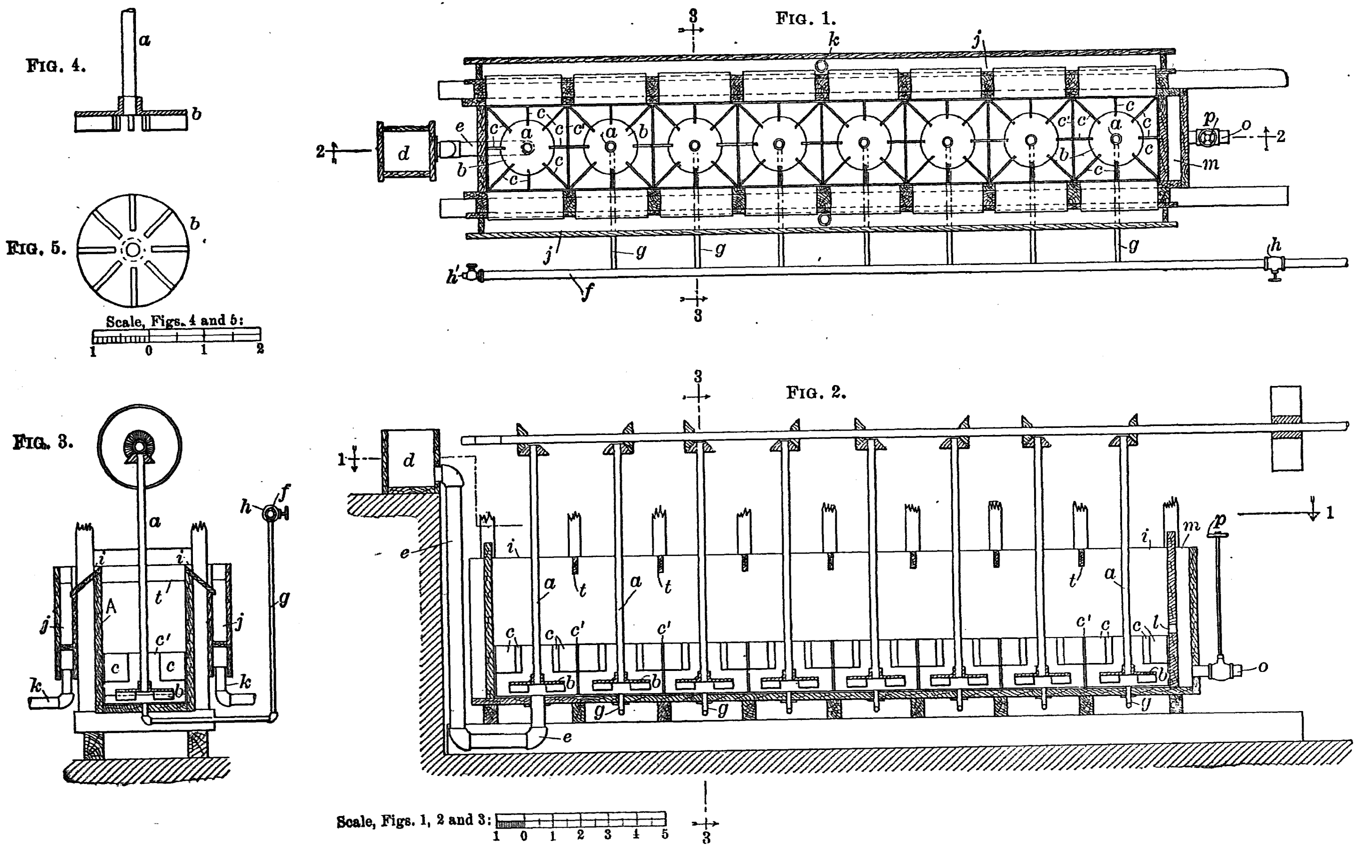


FIG. 38
24-in. Hebbard sub-aëration machine

5 ft. deep, allowing 3 ft. by 3 ft. cross-sectional area for each 24-in. disk. Feed is introduced into the machine either through a feed pipe (*e*) from a pressure box (*d*), under the first disk, or it may be introduced by means of an ordinary feed box through a slot in the end wall. In the machine pictured, air under from two to five pounds pressure per square inch is supplied through the pipes (*g*) directly under each disk except the first. In machines fed through the end of the trough, air is also supplied under the first disk. Froth overflows the sides of the trough. Tailing is discharged through the slot (*l*) into the box (*m*). Pulp level is regulated by means of the valve in the discharge pipe (*c*). The capacity of a 24-in. machine is about 80 tons of sandy feed per compartment per 24 hours in a pulp containing 20 to 25 per cent. solids, with a power consumption of 10 h.p. per spindle. An 18-inch machine treats about 40 tons per cell with a power consumption of 6½ to 7 h.p. per spindle. The air consumption is about ½ cu. ft. per minute per ton of daily capacity at a pressure of five pounds per square inch. The machine has been almost uniformly unsuccessful in the mills.

The Cascade machine utilizes the action of a stream of pulp plunging into a body of pulp to obtain the aëration necessary for flotation. The pulp fed to such a machine must be premixed with oil. One form of the machine is shown in Fig. 39. It consists of a sheet-iron tank with a cylindrical section about 6 ft. diameter by 1 ft. high and two conical sections of the same base attached thereto as shown. Pulp under a head of 3 to 4 ft. is discharged through the nozzle (*b*) within the perforated pipe (*c*) onto the surface of the pulp within this tank. The froth formed overflows the lip (*e*) and is carried off in the annular launder (*f*). Tailing is discharged through the pipe (*g*). Pulp level is regulated by a valve in this pipe. The dish-shaped casting (*d*) is designed to retard the flow of pulp through the machine. Machines are placed, several in series, with sufficient vertical distance between to give from 2 to 3 ft. head on the discharge nozzle (*b*) and a free fall of from 1 to 2 ft. from the nozzle to the surface of the pulp in the machine. These machines are not yet sufficiently standardized to allow a definite statement of capacities. A measure of capacity

may, however, be obtained by allowing for an average rate of flow through the machine, equivalent to that through the froth-

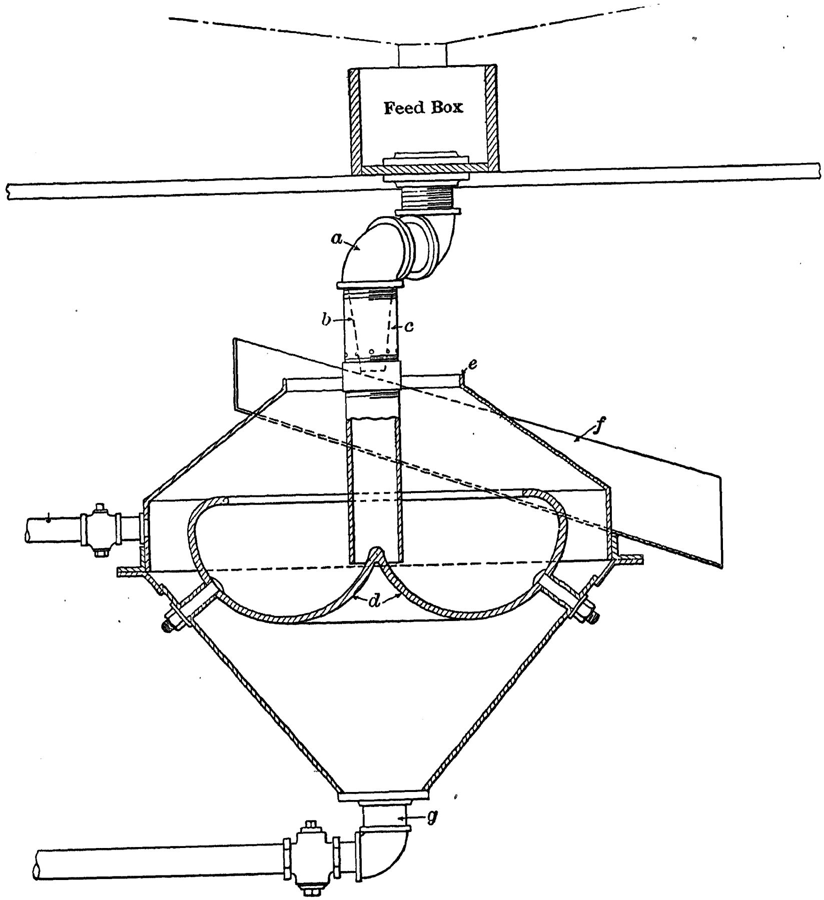
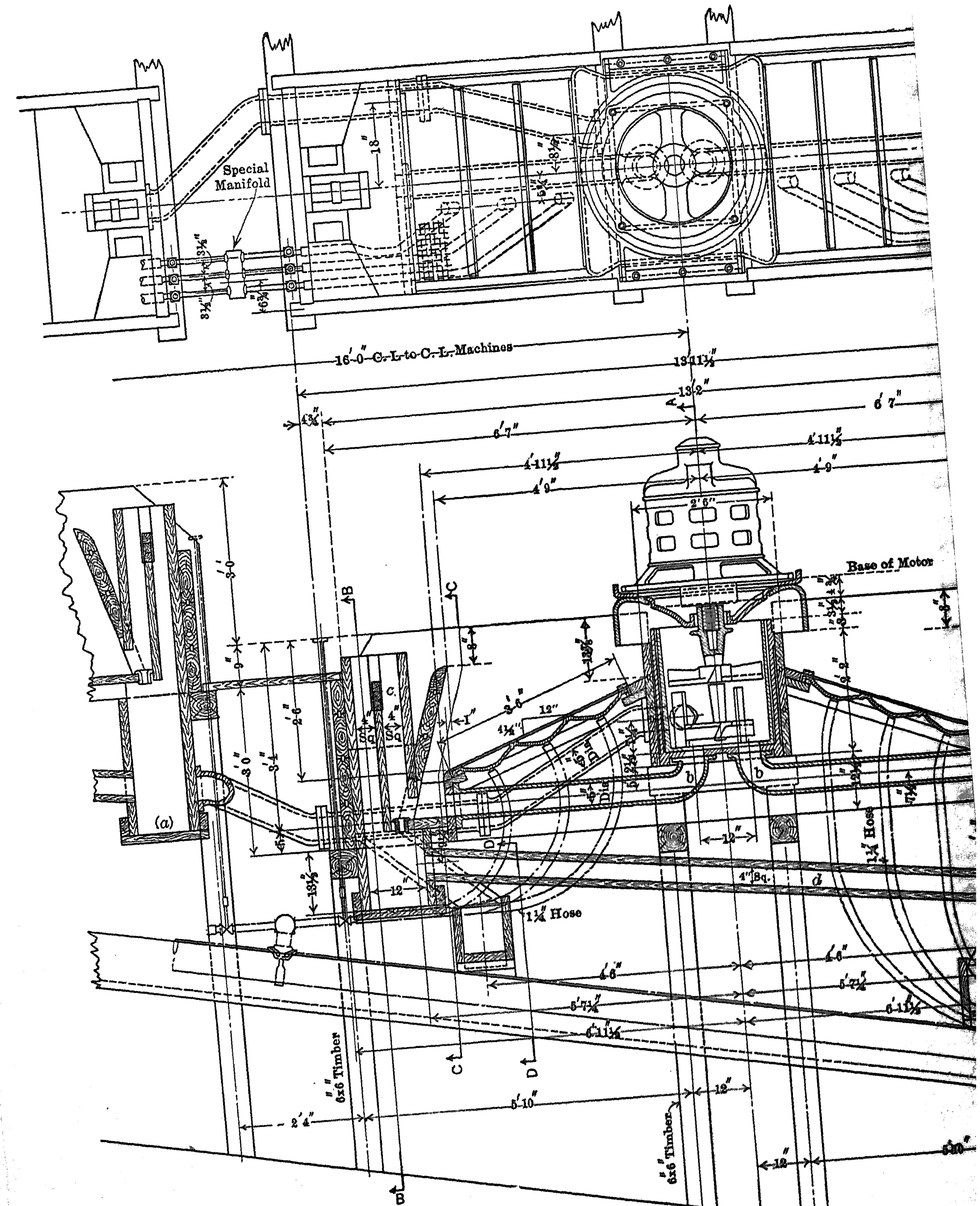


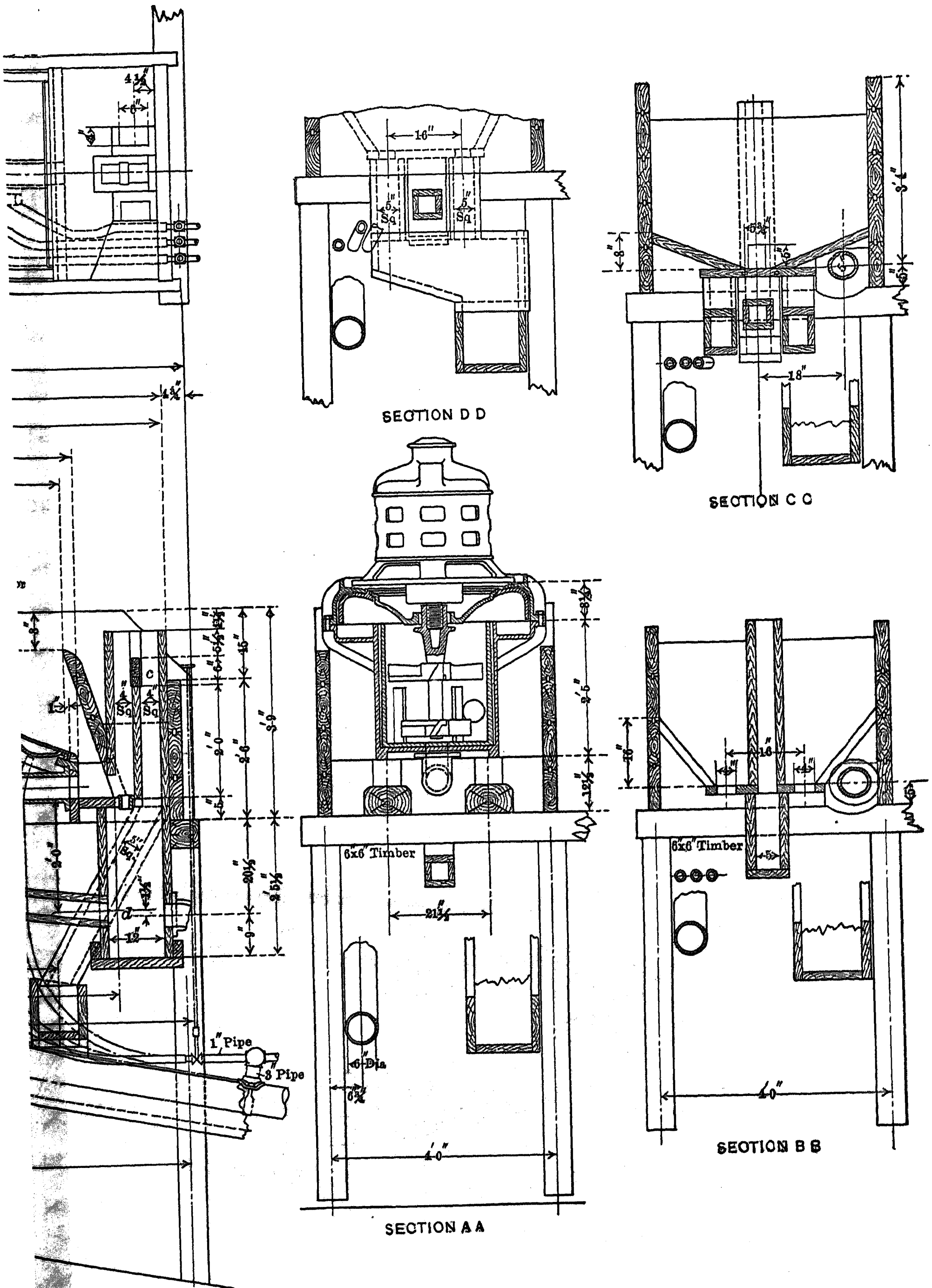
FIG. 39
Court Cascade flotation machine

separating compartment of a Minerals Separation machine. The only power required is that sufficient to elevate the pulp. It is the writer's opinion, however, that a sufficient number of successive treatments is necessary, in order to produce an economical recovery, to necessitate power in the elevation of pulp equal to that consumed in the other bubble-column machines.



F

General arrangement, 24-in.



G. 40
Janney mechanical-air machine

Combination Machines

Several mill flotation machines utilize combinations of the physical phenomena employed in the machines previously described, in order to produce flotation. The following are examples.

The Janney mechanical-air machine utilizes pulp-body concentration and bubble-column concentration. The machine is shown in Fig. 40. The combination is effected by placing air baskets in the froth-separating compartments of a Janney mechanical cell, at the same time reconstructing these compartments to accommodate the baskets. The agitating compartment is the same as that in the Janney mechanical machine with an individual vertical motor. The three-compartmented air baskets in each froth-separating compartment are supplied with air at from four to five pounds pressure on the supply side of the regulating valves. The machines are set up end-for-end as indicated in the drawing. The first machine in series is usually preceded by a Janney emulsifier. Pulp passes from the emulsifier discharge box, which may be taken as represented by box (*a*) in the drawing, through a pipe by gravity into the agitator compartment of the machine and is thrown up over the top of the agitator compartment onto the air baskets. Froth overflows the lips of the air-basket compartments. The tailing is in part circulated through the pipes (*b*) and finally passes through the adjustable overflow slots (*c*) into the tailings discharge launder (*d*) and thence to the following machine. By thus utilizing both methods of froth concentration, it is possible to make a good recovery in a relatively small number (five to six) of machines in series. The capacity of a 24-inch, five-compartment machine on silicious ore in a pulp containing 20 to 25 per cent. solids, is 150 to 200 tons per 24 hours. The power consumption per agitator is six to seven h.p. The air consumption per square foot of air basket is from 5 to 10 cu. ft. of free air per minute at four to five pounds pressure, corresponding to an additional power of five to eight h.p. per machine.

The K and K machine, which is an improvement on the Rork machine, utilizes the principles of both the centrifugal-

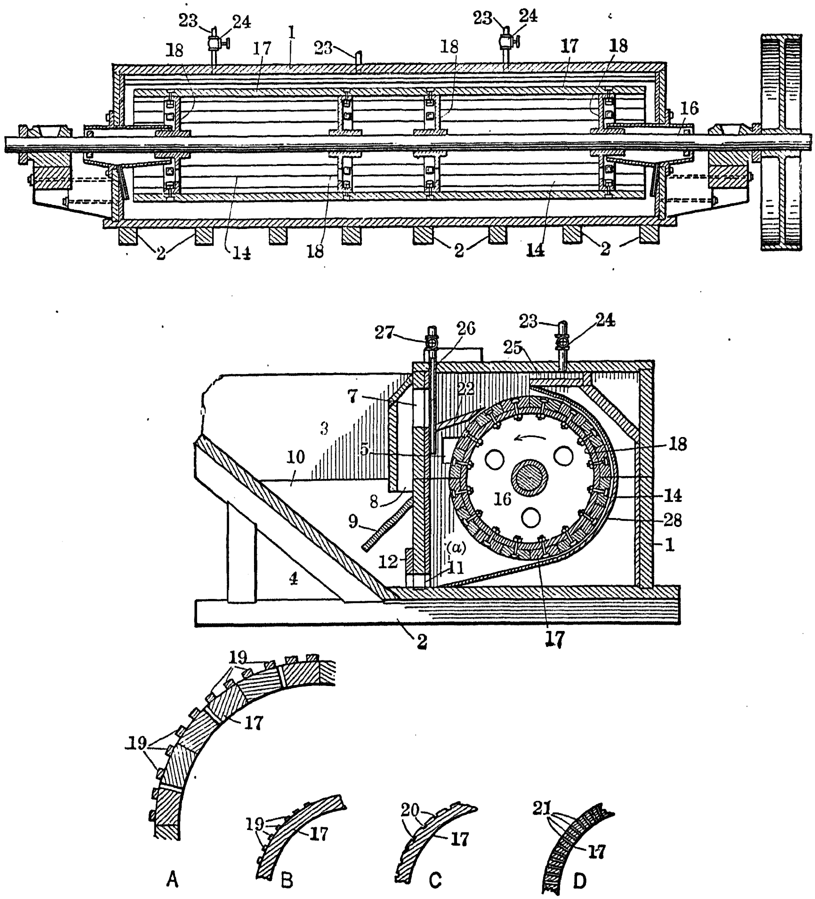


FIG. 41
K and K machin

type and the cascade-type bubble-column machines. A diagrammatic sketch of the machine is presented in Fig. 41, taken from U. S. patent 1,174,737 to F. B. Kollberg and M. Kraut. The essential parts are an aërating compartment (1) and a froth-separating compartment (3). Aëration is accomplished by rapid revolution of the cylinder (17) which is about 30 in. diameter and 9 ft. long. Pulp is introduced at one end of the aërating compartment at a point somewhat above the shaft (16) and

discharges through a pipe at the other end of the settling compartment. The pulp level is maintained well below the center of the shaft in the aërating compartment. Circulation of pulp within the machine is accomplished through the ports (11). Aëration is accomplished differently according to which of the cylinder surfaces, *A*, *B*, *C*, *D*, is used. When *B* and *C* are used air is carried into the pulp on the down coming side in the spaces between the cleats. This is essentially the action of the plunging stream in the Cascade machine. When type *D* is used there is, in addition to air thus carried into the pulp, a creation of vacua in the ports (21) into which air passes from the center of the cylinder. A similar combination of phenomena occurs in the use of *A*. The direction of rotation is as indicated by the arrow. Aërated pulp is thrown through the port (7) and introduced under the hood (8) over the baffle (9) below the surface of the pulp in (3). Froth overflows the lip of compartment (3). Air is allowed to enter through pipes (23) and water can be added through pipes (26).

The first machine of this type was apparently that patented by C. E. Rork in 1915 (U. S. patent 1,136,485). This device had 12 arms bolted to a central shaft to form a typical paddle wheel. These arms did not extend the full length of the shaft but were broken to form a series of five or six paddle wheels on the same shaft, these wheels operating in compartments in a common box and each compartment was discharged into corresponding froth-separating compartments. Due to the considerable pressure exerted on this paddle shaft, both on account of the shape of the paddles and the structural features, the shaft bent so badly at high speeds that the device could not be used. In the present form, in which the paddle arms are at most 2 to 3 in. long in a radial direction and in which the paddles are multiplied to such an extent as to form practically a continuous cylinder, the resistance offered by the pulp is materially lessened and the rotating part offers much more resistance to deflection.

The K and K machine is ordinarily driven at from 180 to 200 r.p.m. The capacity varies from 50 to 100 tons per 24 hours

with a power consumption of from 5 to 10 h.p. according to the volume of pulp passed.

Comparison of Machines

The Janney machine is the best of the agitation-type machines, both structurally and from the point of view of metallurgical results. It will probably return sufficiently more metal as compared with the Minerals Separation machine to more than pay for the additional first cost and for any additional power consumption per ton of ore treated. Machines of the modified Callow type such as the Miami and Inspiration are more economical of floor space than those of the typical Callow type and will also be cheaper in first cost, if the tonnage to be treated is sufficient to warrant their use. Metallurgical results in both classes of machines will be about the same. It is probable that the highest possible metallurgical results can be obtained in the Janney mechanical-air machine, but at an expenditure of power which is unwarranted in most cases. Machines of the K and K type are attractive from the point of view of low power consumption but they are not economical of floor space in case of large installations, and are subject to considerable mechanical difficulties. It is significant that in several cases these machines have been displaced, after trial, by machines of other types, in spite of the lower power consumption. Machines of the pneumatic variety show a lower upkeep and a lower cost of operation over considerable periods of time than do machines of the agitation type. It is the writer's opinion that ten years' time will see the agitation-type machine almost completely displaced by machines of the pneumatic type. The Hebbard sub-aëration machine has given much trouble mechanically and has been thrown out of many mills after trial on this account. The Ruth and Groch machines have not been given enough trial to warrant general statements as to their usefulness. Much is claimed for machines of the Cascade type but, notwithstanding their trial at many of the larger mills in the country, no installations of any size have been made. Installations of this machine will be awkward from the point of view of mill construction and opera-

tion. Oil-flotation machines are nowhere in use today and skin-flotation machines are used in but few places. As previously stated, the use of the latter type of machine is justified only where a high-grade concentrate must be made, irrespective of the grade of the tailing

Flotation Flow-sheets

Typical flotation flow-sheets may be classified into two general types on the basis of the part which flotation plays in the mill treatment scheme and each of these classes may be again subdivided on the basis of the method of routing the pulp through the flotation machines. On the first basis, a flow-sheet is of the "primary" type when flotation is the primary or principal means of concentration employed and the bulk of the concentrate is recovered thereby. The flow-sheet is of the "secondary" type when flotation is an accessory or subordinate process and some other means of concentration, usually gravity concentration, is the principal method of treatment. On the basis of pulp routing, a flow-sheet is of the "concentrate-middling" type when the flotation feed pulp passes through a set of machines in series, and these machines deliver finished froth concentrate off the early cells, a clean tailing as the underflow or spigot product of the last cell; and a low-grade froth or middling as the overflow of the later cells, this middling being returned to the head of the machine. A flow-sheet is of the "rougher-cleaner" type when two machines, not in series, comprise the flotation installation, and the first machine makes a finished tailing and a low-grade concentrate which latter is sent to a second machine for cleaning. The second machine makes a finished high-grade concentrate and an underflow or spigot product constituting a middling, which is returned to the first or rougher cell. Combinations of these two methods of routing are also met with and may be classed as "combination" methods. Figures 42, 43 and 44 present, in a general way, these three routings. It will be understood, of course, that many combination routings are possible, although most of them are of the general character shown. It will thus be seen that mill flow-sheets may

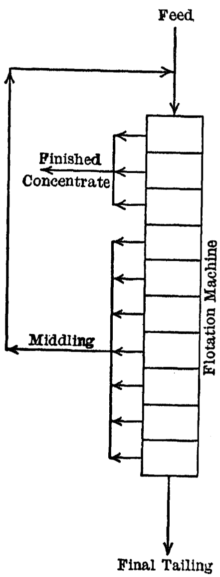


FIG. 42
Concentrate-middling routing

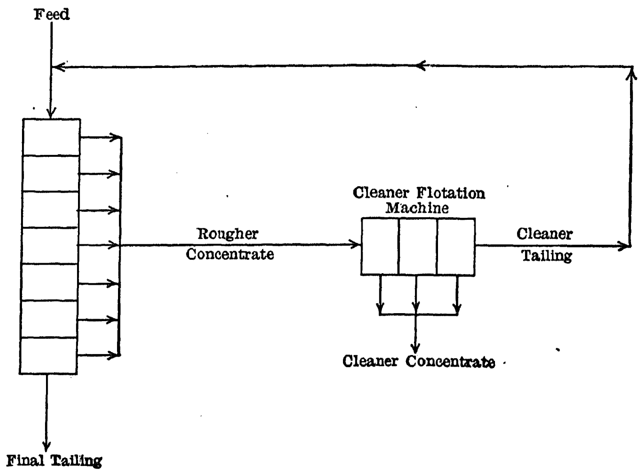


FIG. 43
Rougher-cleaner routing

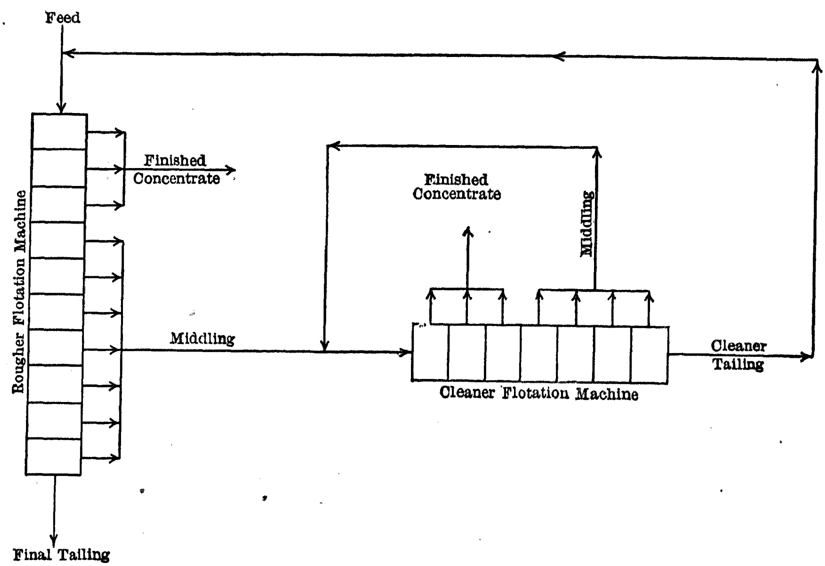


FIG. 44
A combination routing

e classified under six possible heads: (1) primary, concentrate-iddling routing; (2) primary, rougher-cleaner routing; (3) pri-ary, combination routing; (4) secondary, concentrate-middling uting; (5) secondary, rougher-cleaner routing; (6) secondary, mbination routing.

The bases for the differences in methods of flotation treat-ent are (1) the differences in the mode of occurrence of the aluable minerals in an ore and (2) the inability inherent in otation processes to make a finished concentrate and a finished iling with no material of intermediate value, in one treatment n one and the same machine. If the sulphide mineral in an ore ccurs in coarse aggregates, a considerable proportion can be ved by gravity concentration at a less cost than by flotation, nd ordinarily in the form of a concentrate that is more valuable an the concentrate made by flotation; assuming, of course, at the specific gravity of the gangue is sufficiently different om that of the sulphide to make gravity concentration efficient. n such case, flotation will probably form a subordinate part of e flow-sheet. On the other hand, if the sulphide mineral is isseminated through the ore in fine grains and the difference specific gravity between the sulphide mineral and the gangue not great, gravity concentration can recover only a small part f the valuable mineral and flotation should form the principal art of the treatment scheme. The choice as to the method of uting depends, to a large extent, on the percentage of floatable inerals present in the ore and on the grade of concentrate esired. A rougher-cleaner routing is usually used where the ercentage of mineral in the flotation feed is low, and the con-entrate-middling routing or combination routing is used when e percentage of mineral is high. The rougher-cleaner routing best adapted to making a high-grade concentrate.

The following tabulation shows the character of the flow-eets, on the basis of the classification presented, at represen-tive concentrating plants. Several of them are given in detail n subsequent pages.

Primary.

Rougher-cleaner routing.

Miami Copper Co. (copper).

Inspiration Copper Co. (copper).

Arizona Copper Co. (copper).

National Copper Co. (copper).

Concentrate-middling routing.

Mountain Copper Co. (copper).

Utah Leasing Co. (copper).

Combination routing.

Consolidated Arizona Smelting Co. (copper).

Anaconda Copper Mining Co., Zinc plant.

Secondary.

Rougher-cleaner routing.

Daly Judge Mining Co. (lead-zinc).

Burro Mountain Concentrator, Phelps, Dodge Corporation (copper).

Magma Copper Co., Copper-sulphide plant.

Magma Copper Co., Zinc plant.

Concentrate-middling routing.

Anaconda Copper Mining Co., Copper plant

Silver Peak Mill, N.S.W. (silver-lead).

Combination routing.

Timber Butte Mill (zinc).

Bunker Hill and Sullivan, West Mill No. 2 (lead)

Silver King Coalition (silver-lead).

Federal Lead Co., No. 4 mill (lead).

Miami Copper Co. mill, Miami, Ariz. This mill consists of six sections of variable flow-sheet. One type is shown in Fig. 45. The ore is chalcocite in a gangue of decomposed granite and schist. It contains approximately 1½ per cent. sulphide copper and from 0.2 to 0.3 per cent. carbonate copper. The capacity of a section is from 800 to 1000 tons per 24 hours. Pulp is fed to the flotation machines with a consistency of 20 to 25 per cent. solids. From 0.5 to 0.75 lb. per ton of an oil mixture consisting of 90 per cent. coal-tar oil and 10 per cent. steam-distilled pine oil, is usually added. The pulp is neutral or very slightly alkaline. A concentrate containing from 40 to 50 per cent. copper and a tailing containing from 0.1 to 0.15 per cent. sulphide copper is made.

The Inspiration Consolidated Copper Co. mill, at Miami, Ariz., is described by R. Gahl in Trans. A. I. M. E., Vol. LV, page 576. The mill employs both agitation-froth flotation and bubble-

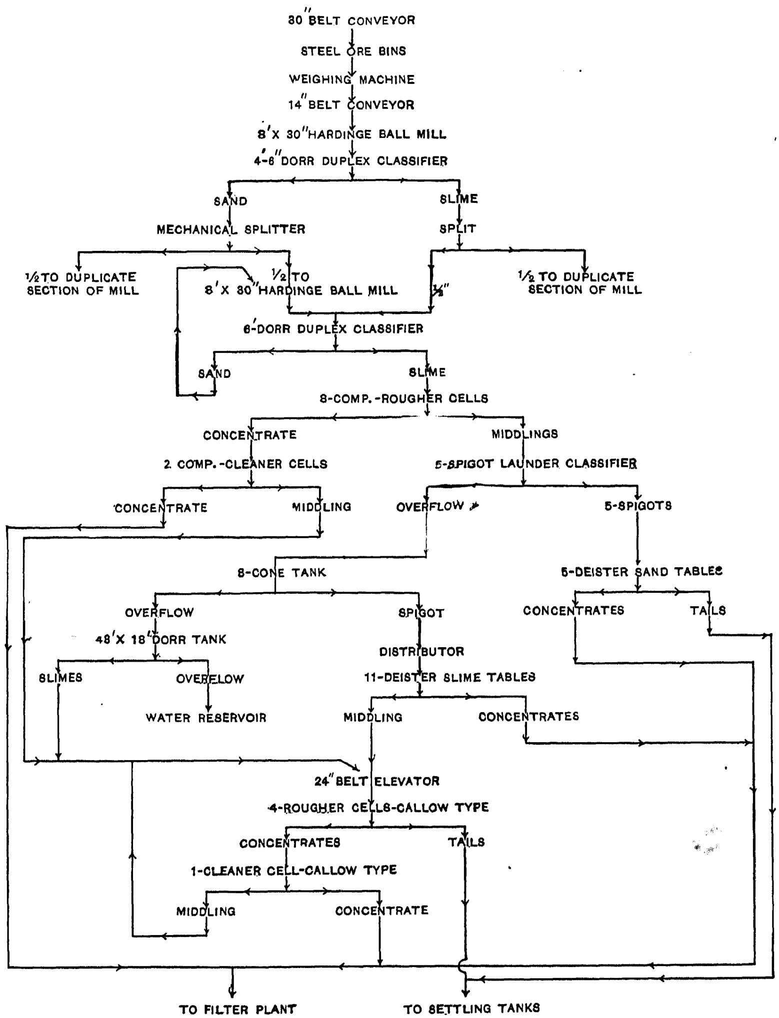


FIG. 45

Flow-sheet, Miami Copper Co. Primary flotation-unit No. 5, 800 to 1000 tons per 24 hr.

column notation, some of the sections being equipped with one type of machine and some with the other. The bulk of the concentration is done by pneumatic-type machines. Of these machines some of the sections employ the Callow type and some the Inspiration type. Flow-sheets of sections employing the

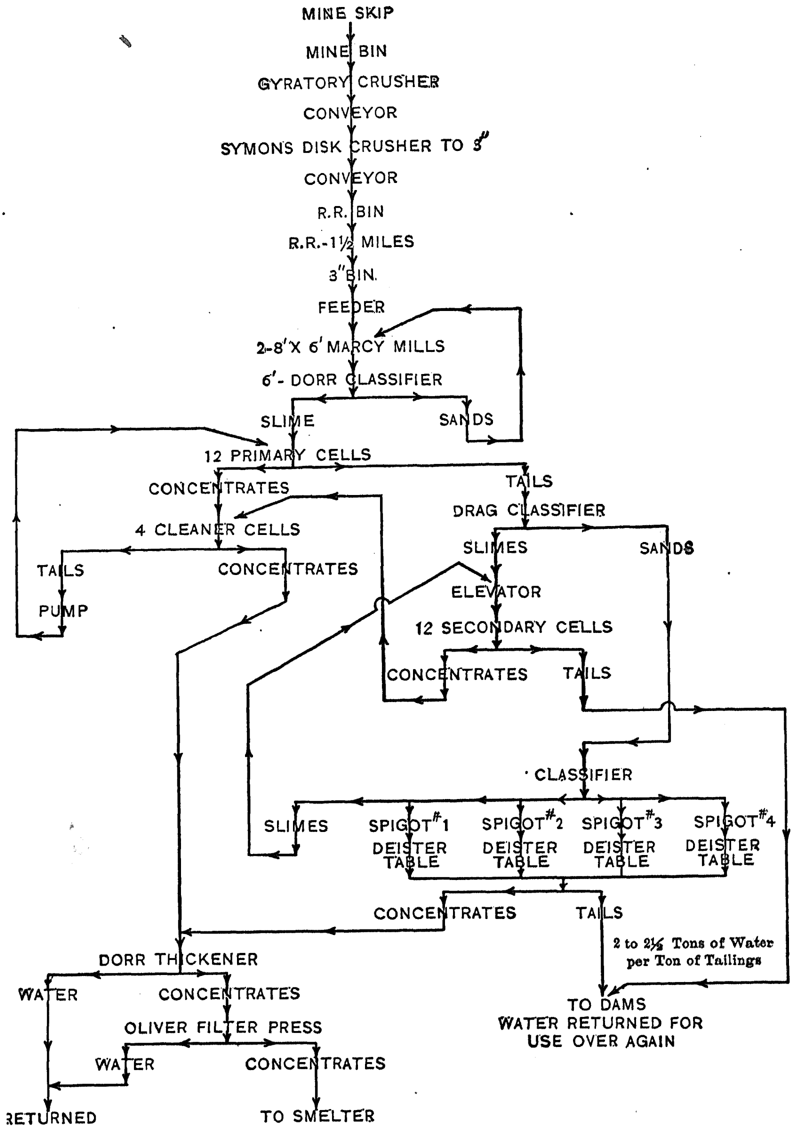


FIG. 46

Flow-sheet, Inspiration Consolidated Copper Co. Callow cell section

two types of machines are given in Figs. 46 and 47. The ore is chalcocite with some oxides and carbonates of copper in a gangue of schist and decomposed granite. The rated capacity

of a section is 800 tons per 24 hours. The pulp is fed at approximately 25 per cent. solids. The ore contains in the neighborhood of 1 per cent. sulphide copper and a recovery of approximately

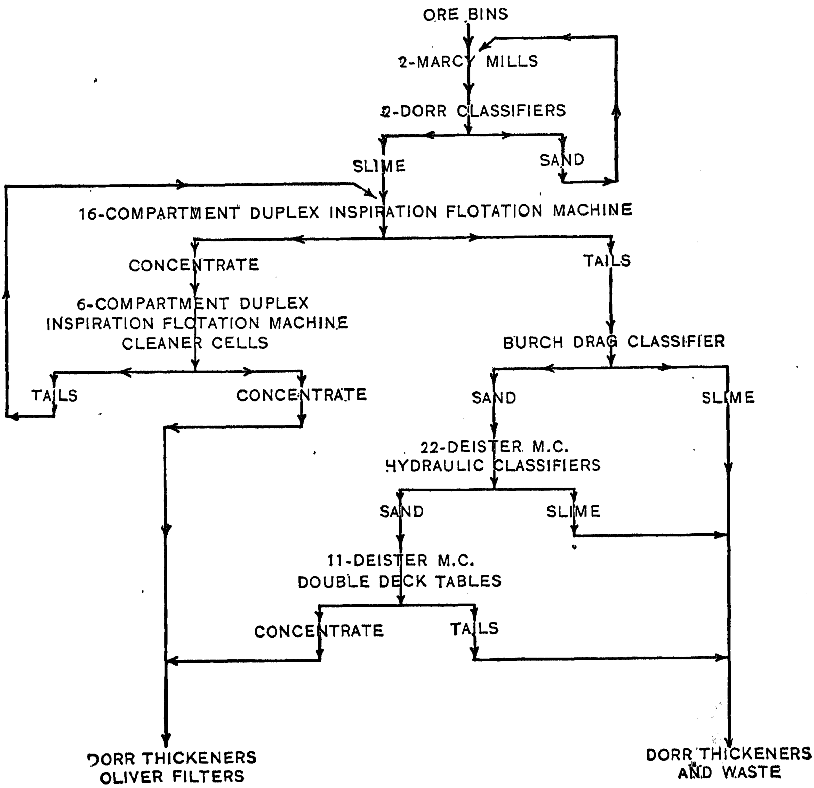


FIG. 47

Flow-sheet, Inspiration Consolidated Copper Co. Inspiration cell section

90 per cent. is made. A mixture of 95 parts of coal tar and five parts of wood oil in a neutral pulp is used. Dr. Gahl gives the oil consumption (1916) at about 1½ lb. per ton of ore. It is probable that it is less at the present time. The following construction data are given in Dr. Gahl's paper. The compartments in the Inspiration rougher machines are 3 ft. by 3 ft. 4 in. and in the cleaner machines three ft. by three ft., giving a combined compartment area of 516 sq. ft. and a capacity per square foot, on the basis of 800 tons, of 1.55 tons per 24 hours.

It is stated that 100 per cent. overload did not seriously affect the grade of the tailing. The area of porous bottom in one of these machines is 340 sq. ft. The air consumption per sq. ft. of porous bottom is 11.8 cu. ft. per minute at a pressure of $4\frac{1}{2}$ lb. per sq. in. at the blower. The power required to deliver this air is 2.63 K.W. hours per ton treated, on the 800-ton basis. On the same basis the following figures hold: 0.425 sq. ft. of porous surface per ton per 24 hours; 0.645 sq. ft. of flotation machine area per ton per 24 hours; 1.75 sq. ft. of floor space devoted to flotation per ton per 24 hours; 5.68 sq. ft. of floor space per ton per 24 hours in the concentrator proper, excluding coarse crushing; 13.3 sq. ft. of floor space per ton per 24 hours for concentrator proper and settling department.

Mountain Copper Company, No. 1 concentrator at Minnesota Station, Shasta County, Cal., is described by L. C. White in Mining and Scientific Press, Sept. 6, 1919. The flow-sheet is shown in Fig. 48. The ore is chalcopyrite and pyrite in an alaskite-porphyry gangue. It carries 2 per cent. copper and 8 per cent. iron. The mill has a capacity of 550 tons per 24 hours, crushing to less than 4 per cent. on 60-mesh. The consistency of the pulp going into the flotation machines is about 30 per cent. solids. The oil mixture consists of 50 parts kerosene acid sludge and 50 parts of a mixture of crude turpentine, tar oil, and light fuel oil. Eight-tenths of a pound per ton of this mixture is fed to the grinding mills. A concentrate containing 15 per cent. copper is made with an average recovery of 92 per cent. and a 7 : 1 ratio of concentration.

Consolidated Arizona Smelting Co. mill, at Humboldt, Ariz., is described by G. M. Colvocoresses in Engineering and Mining Journal, July 14, 1917. The flow-sheet is shown in Fig. 49. The ore consists of chalcopyrite and pyrite in schist and carries 3.2 per cent. copper and 15 per cent. iron. The flotation feed carries about 2 per cent. copper. Flotation concentrates assay 15 per cent. copper, 25 per cent. iron, and 20 per cent. insoluble. The roughing-table concentrate carries about $4\frac{1}{2}$ per cent. copper and 38 per cent. iron. Flotation and table concentrate mixed, forming the mill concentrate, assay 9 per cent. copper, 30 per

cent. iron and 16 per cent. insoluble. Flotation tailing assays about 0.4 per cent. copper. The recovery by flotation is about

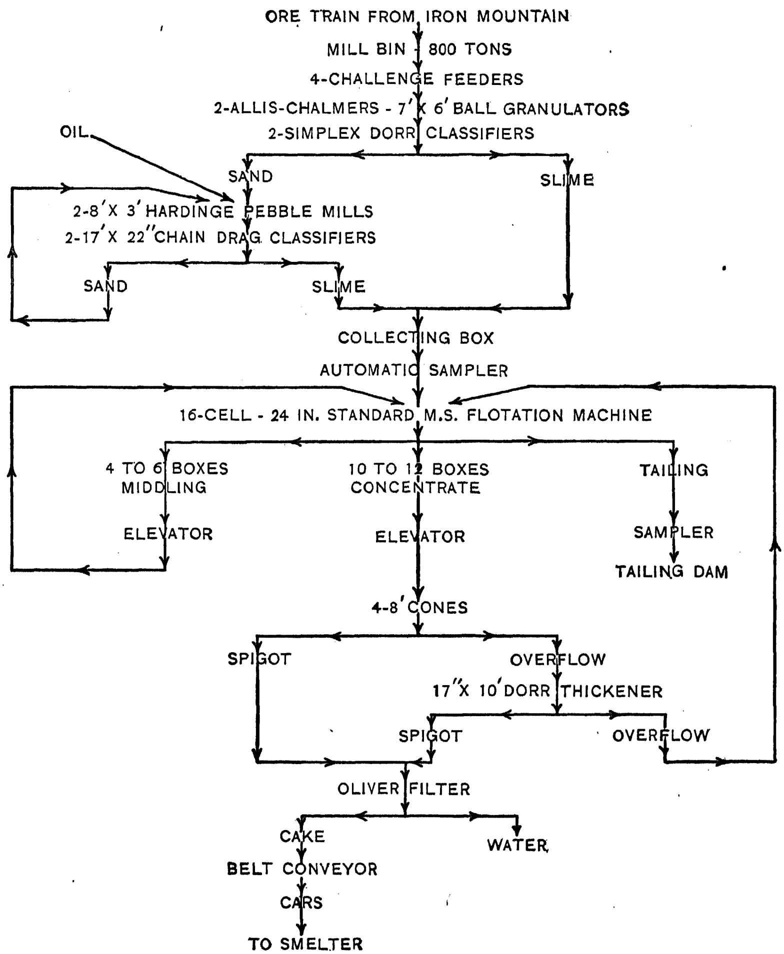


FIG. 48
Flow-sheet, Mountain Copper Co. No. 1 concentrator

82 per cent. The capacity of the plant as shown is approximately 400 tons per 24 hours. The oil mixture consists of 70.4 parts of Standard Oil Company stove oil, 27 plus; 24.9 parts of refined wood creosote; and 4.7 parts Standard Oil Company

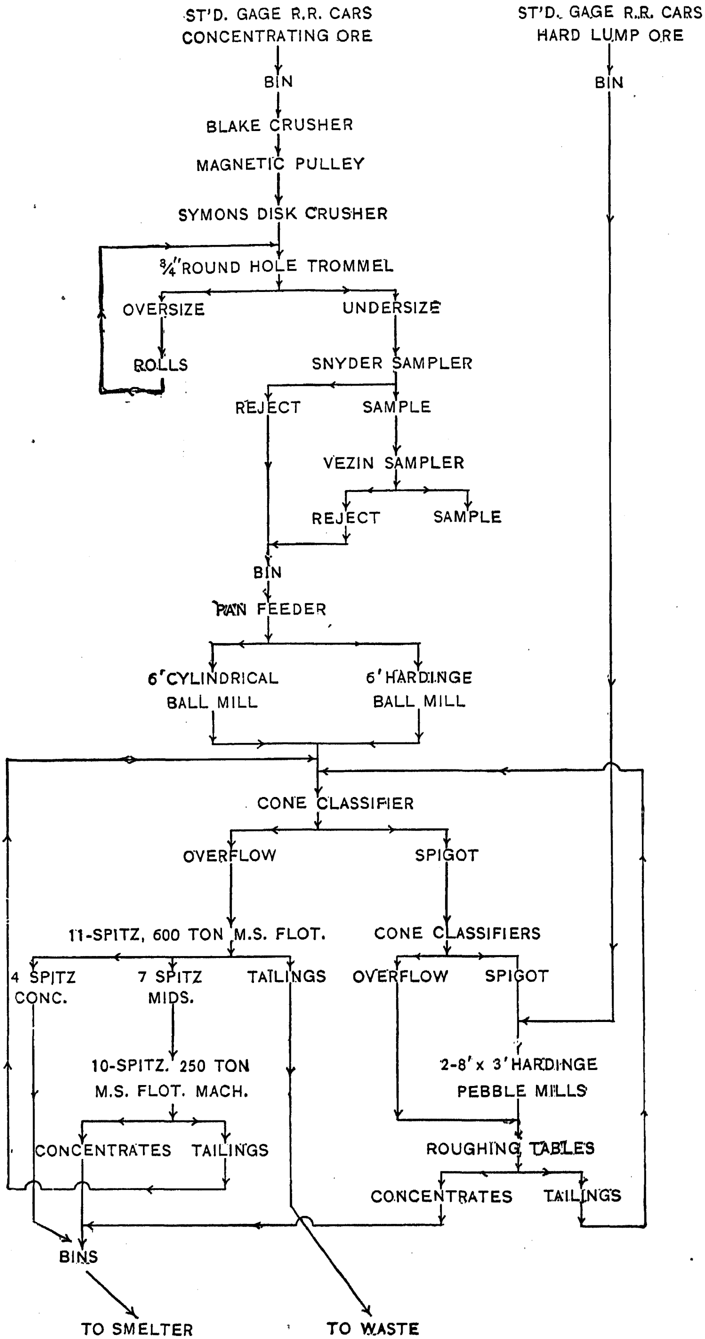


FIG. 49
Flow-sheet, Consolidated Arizona Smelting Co

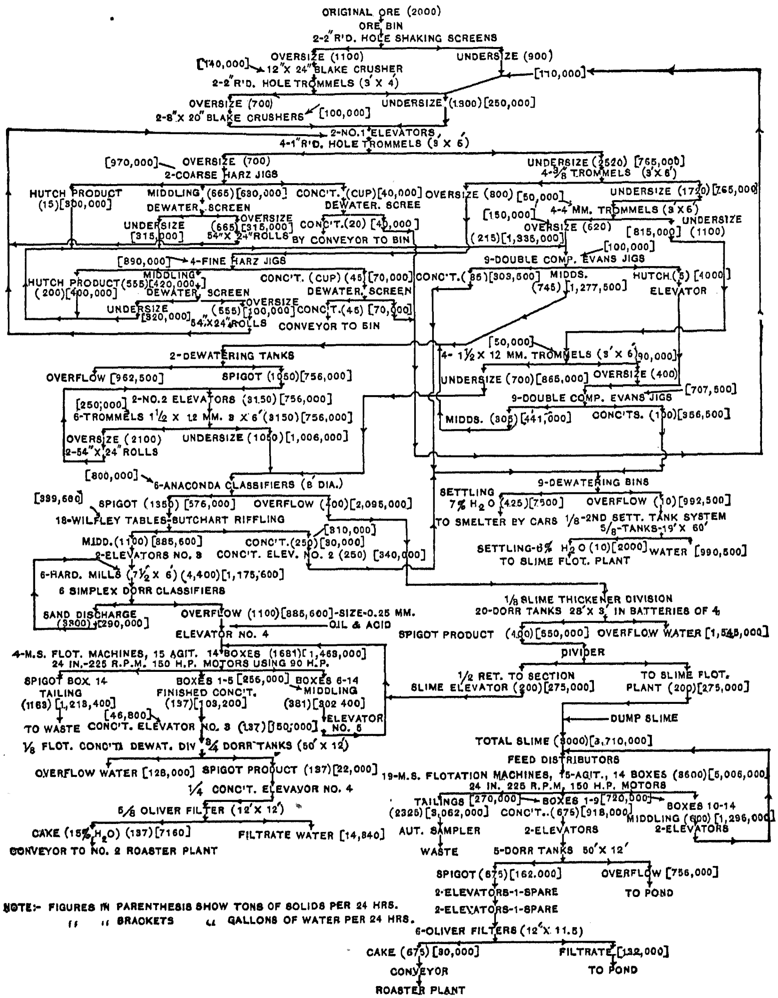


FIG. 50
Flow-sheet, Anaconda copper concentrator

Calol fuel oil, 24 plus. The average consumption of this mixture is 1.31 lb. per ton.

The Anaconda Copper Mining Company operates one concentrator for the treatment of copper sulphide ores and another for the treatment of zinc sulphide ores. The flow-sheets of the two plants are given in Figs. 50 and 51 respectively, as represented

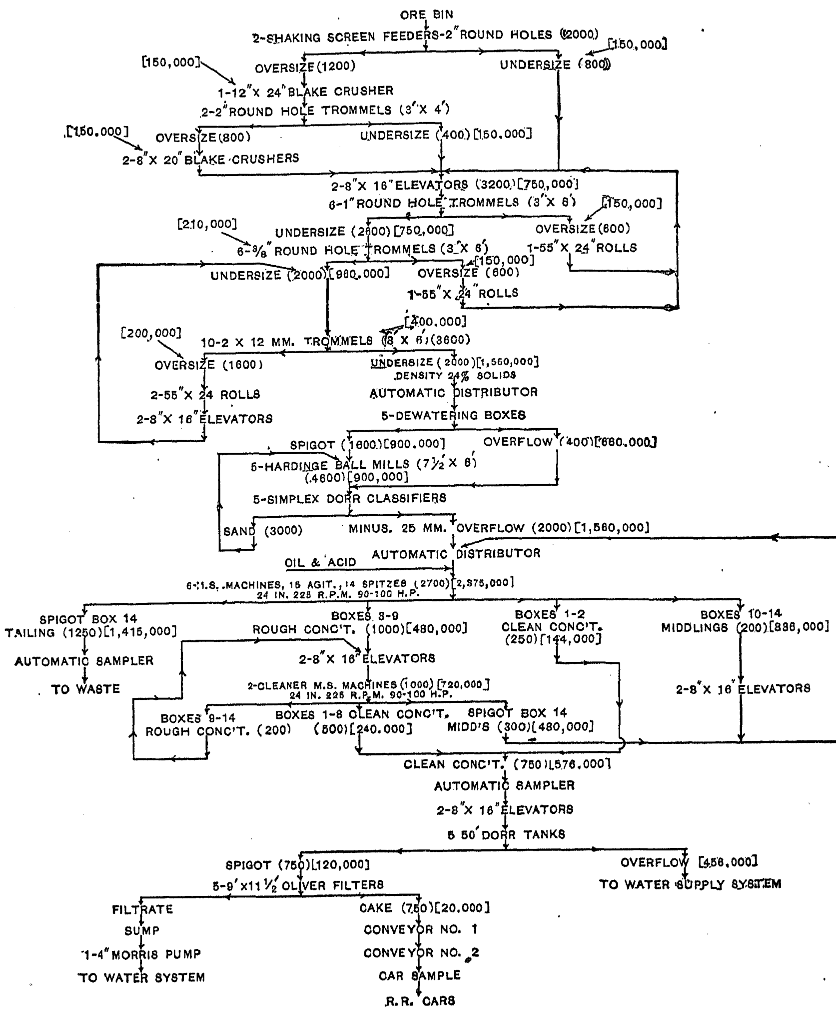


FIG. 51

Flow-sheet, Anaconda zinc ore concentrator, 2000 ton unit

Note:—Figures in parenthesis show tons of solids per 24 hours. Figures in brackets show gallons of water per 24 hours.

Density Per Cent Solids	Per Cent
Pulp from Crusher Section	24
Total Feed to Hardinge Mills	55
Original Feed to Flotation	24
Total Feed to Flotation	22
Total Concentrate	25
Feed to Oliver Filters	61
Filter Cake	90

by A. E. Wiggin in the trial of the case of Minerals Separation, Ltd. *vs.* Butte and Superior Mining Company, in the district court of Montana. The copper ore carries about 2.9 per cent. copper and considerable pyrite. Concentrates carry 8 per cent. copper, 21 per cent. iron and 28 per cent. soluble. The recovery in 1916 for the whole mill was 95.45 per cent. and the recovery for flotation about 94 per cent. In the sand-plant flotation, the oil mixture consisted of 3 to $3\frac{1}{2}$ lb. per ton of kerosene sludge acid and about 0.3 lb. per ton of hardwood creosote. From 6 to 8 lb. per ton of sulphuric acid was also added. In the slime plant there was added from 3 to $3\frac{1}{2}$ lb. of kerosene sludge acid, $2\frac{1}{2}$ to 3 lb. wood creosote, and 15 lb. of sulphuric acid per ton of ore. The capacity of the plant is shown by numerals on the flow-sheet.

The feed to the zinc concentrator carried 13.3 per cent. zinc and a concentrate containing 33 per cent. zinc with a recovery of 92.8 per cent. was reported. This extremely low-grade zinc concentrate is allowable for the reason that it is treated electrolytically to recover the zinc, rather than by ordinary zinc-smelting methods. The oil mixture used consisted of 0.7 lb. per ton of kerosene sludge acid and 2.7 lb. per ton of wood creosote. Sulphuric acid to the amount of 22.7 lb. per ton was also used. Tonnage and pulp-density figures are given on the flow-sheet.

Daly-Judge Mill, Park City, Utah, is described by A. B. Parsons in the Salt Lake Mining Review, February 29, 1916. The flow-sheet is given in Fig. 52. Feed to the mill is a zinc-lead ore carrying 6 per cent. zinc, $3\frac{1}{2}$ per cent. lead and 5 per cent. silver. The flotation concentrate is separated on tables into a zinc concentrate carrying 48 per cent. zinc, 6 per cent. lead, 4 per cent. iron, 25 oz. silver and $3\frac{1}{2}$ per cent. silica; and a lead-iron concentrate carrying 42 per cent. lead, 7 per cent. zinc, 16 per cent. iron, 35 oz. silver and $1\frac{1}{2}$ per cent. silica. The oil used is a wood creosote. The capacity of the plant is given as approximately 50 tons per 24 hours. It would look from the flow-sheet as though the actual capacity would be considerably above this rated figure, especially in so far as the flotation equipment is concerned.

The Burro Mountain Concentrator of the Phelps, Dodge Corporation is described by F. C. Blickensderfer in the Engineering and Mining Journal, July 14, 1917. He gives the flow-sheet

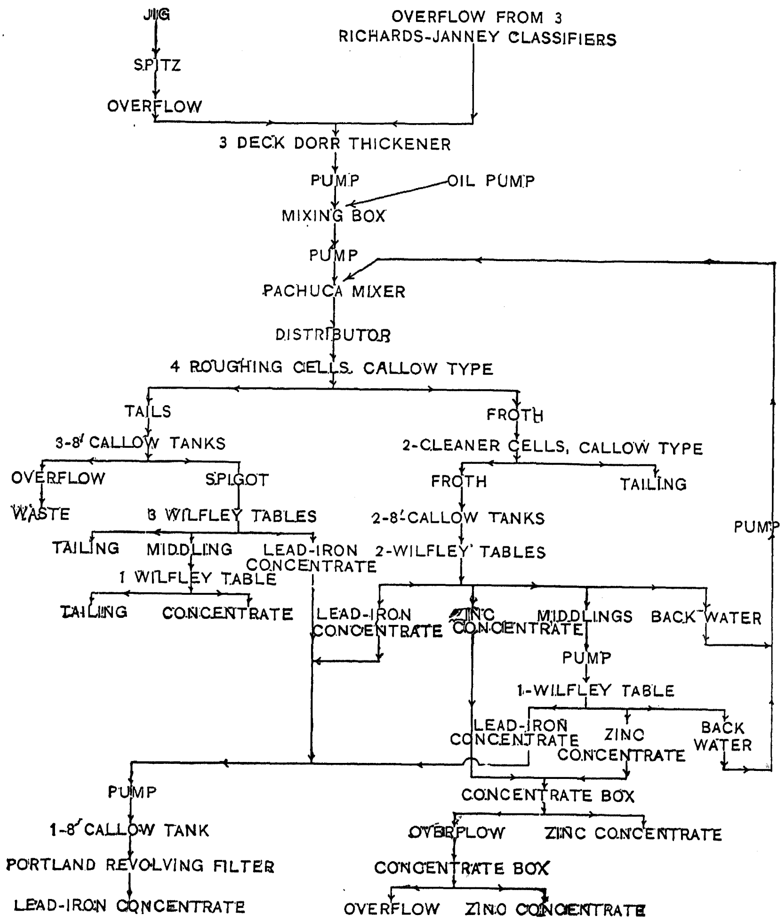


FIG. 52
Flow-sheet, Daly-Judge Mill, Park City, Utah

shown in Fig. 53. The ore assays about 1.9 per cent. copper of which 0.2 per cent. is carbonate and oxide and the balance chalcocite. The gangue is porphyry. A recovery of approximately 77 per cent. on the sulphides is reported in Mr. Blickensderfer's paper. The flotation machines are of the Kraut and Kollberg

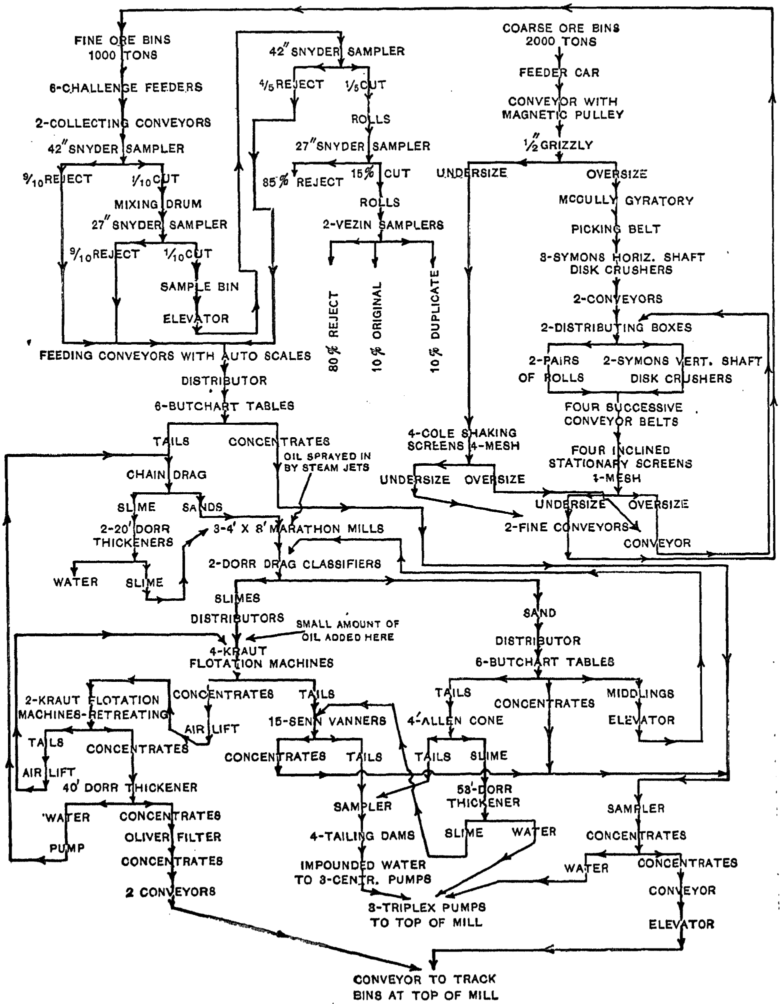


FIG. 53

Flow-sheet, Burro Mountain Concentrator

type and are fed at the rate of 75 tons per 24 hours in a pulp containing 20 per cent. solids. Oil to the amount of 2.75 lb. per ton of ore is added at the points shown. Lime is added at the crushing machines to protect the iron work from attack by soluble salts. The ore coming from one part of the mine re-

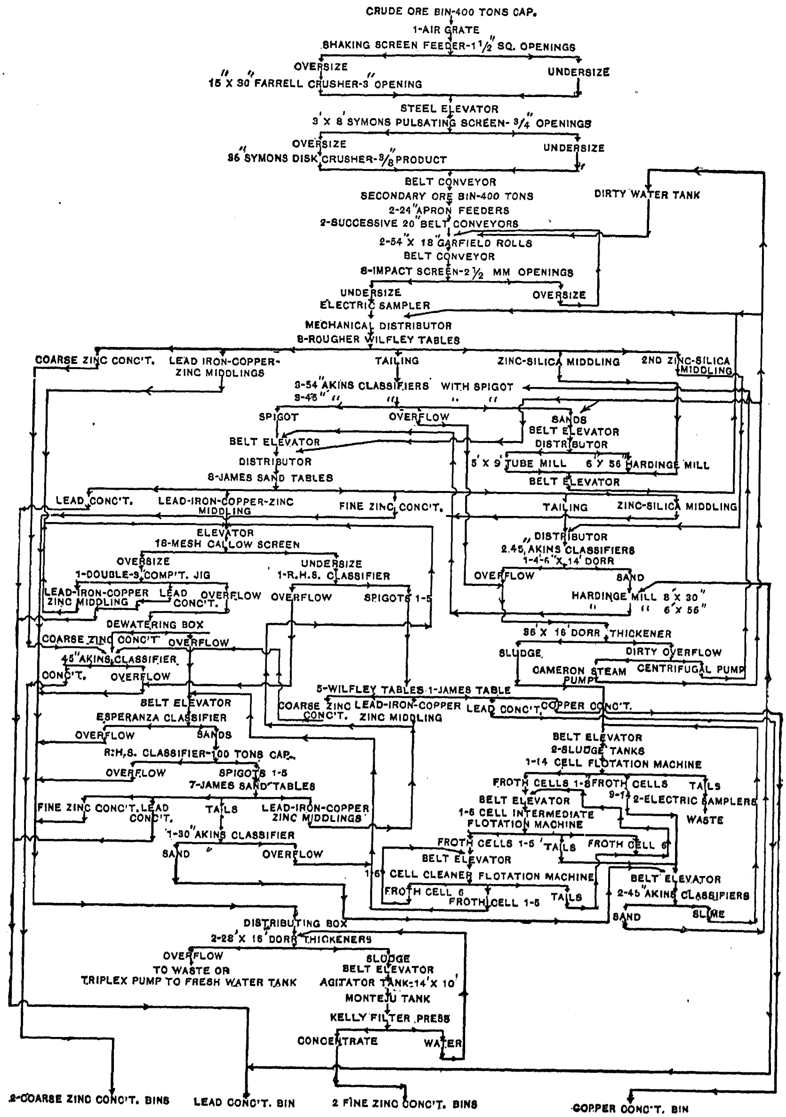


FIG. 54
Flow-sheet, Timber Butte Milling Co.

quires the addition of about eight pounds of sulphuric acid per ton at the flotation machines. The other ores are treated with lime as the only inorganic agent present.

Timber Butte Milling Company, Butte, Montana. The operations at this mill were described by W. N. Rossberg in the trial of the suit of Minerals Separation, Ltd., *vs.* Butte and Superior Mining Company, in the district court of Montana, and the flow-sheet shown in Fig. 54 was presented by Mr. Rossberg. The capacity of the plant as shown is 450 to 550 tons per 24 hours. The ore consists of sphalerite and galena in a granitic gangue. The metallurgical results for the years 1915, 1916 and part of 1917 are given in Table XII. Tables XIII and XIV are of interest as showing the changes in oil and acid feed which are characteristic of much flotation practice. It should be borne in mind in reading the statements as to character of oil and oil consumption in connection with the other flow-sheets herein given and those published in the technical journals, that some such situation as is presented here undoubtedly has existed in most of the other plants and that the statements as to kind and quantity of oil used represent only the most common practice or the practice which happened to obtain at the time the article was written. Tables XIII and XIV illustrate clearly the statement made in another part of this book, that the slogan of "a different oil for every ore" which has been the rallying cry of some so-called flotation experts, needs considerable revision.

TABLE XII

TIMBER BUTTE MILLING CO., BUTTE, MONTANA

Comparison of Yearly Metallurgical Results. Figures Based on Mill Weights and Assays

Date	Original mill feed		Gravity concentrates			Flotation feed	
	Tons	Assay per cent. zinc	Zinc conc. tons	Copper-iron conc. tons	Lead conc. tons	Tons	Assay per cent. zinc
Year 1915	160 461.442 5	16.723	10 694.423 5	1 389.850 0	340.330 0	148 036.839 0	14.377
Year 1916	201 455.030 5	13.385	12 119.840 5	975.720 5	681.710 0	187 677.759 5	11.232
Jan., Feb., Mar., 1917	41 956.168 0	17.426	3 517.486 0	287.126 0	158.729 5	37 892.826 5	14.295
Total, Y'rs 1915–16 and Jan., Feb., and Mar., 1917	403 872.641 0	15.131	26 331.750 0	2 652.696 5	1 180.769 5	373 707.425 0	12.829

Date	Flotation conc.		Total mill tailings		Recovery of zinc in original mill feed				Estimated recovery of zinc in flotation feed	
	Tons	Assay per cent. zinc	Tons	Assay per cent. zinc	Per cent. zinc recovered in fine and coarse zinc conc's. by weight and assays of feed and conc's.	Per cent. zinc recovered in fine and coarse zinc conc's. by tailing and conc's.	Per cent. zinc recovered in all conc's. by weights and assays of feed and conc's.	Per cent. zinc recovered in all conc's. by tailing and conc.	Per cent. zinc recovered in flotation conc's. by flotation feed and conc's.	Per cent. zinc recovered in flotation conc's. by tailing and flotation conc's.
Year 1915	39 486.339 0	51.359	108 550.500 0	1.076	94.52	94.46	95.71	95.55	94.63	94.56
Year 1916	39 140.182 5	50.590	148 537.577 0	0.768	94.17	94.66	95.26	95.75	93.94	94.55
Jan., Feb., Mar., 1917	9 742.600 0	54.474	28 250.226 5	0.714	97.17	96.15	98.31	97.27	97.72	96.34
Total, Y'rs 1915–16 and Jan., Feb., and Mar., 1917	88 369.121 5	51.362	285 338.303 5	0.880	94.68	94.75	95.82	95.88	94.67	94.76

TABLE XIII

Timber Butte Milling Co., Butte, Mont.

Flotation Oil Consumptions—and Acid Consumptions

Date	Flotation oil		Flotation acid		Oils used. Timber Butte Milling Co. Numbers.*
	Pounds per ton of original feed	Pounds per ton of flotaton feed	Pounds per ton of original feed	Pounds per ton of flotation feed	
August 1914.....	0.54	0.60	9.77	10.84	No. 6, 140
September "	0.49	0.54	9.18	10.15	No. 6, 10, 140
October "	0.51	0.56	9.03	9.99	No. 6, 10, 140
November "	0.68	0.78	9.41	10.73	No. 6, 10
December "	0.76	0.82	8.61	9.29	No. 6, 10
5 months Year 1914..........	0.61	0.66	9.38	10.17	
January 1915.....	0.81	0.87	7.70	8.36	No. 6, 10
February "	0.69	0.73	5.56	5.89	No. 6, 10, 12, 50, 90
March "	0.71	0.75	5.11	5.43	No. 6, 10, 12, 52, 88, 90, 160
April "	0.85	0.94	6.45	7.13	No. 6, 10
May "	0.85	0.93	8.17	8.87	No. 6, 10, 87
June "	0.65	0.70	8.69	9.38	No. 10, 87
July "	0.84	0.90	17.06	18.22	No. 6, 10, 87, 89, 171, 174
August "	0.83	0.90	12.91	13.86	No. 10, 89, 174, 192
September "	1.40	1.49	15.94	17.03	No. 10, 156, 174, 191, 192
October "	1.66	1.82	8.82	9.66	No. 170, 174, 191, 192
November "	1.10	1.19	8.16	8.79	No. 6, 21, 23, 87, 110, 174, 192
December "	1.34	1.46	9.67	10.53	No. 6, 21, 23, 87, 110, 192
Year 1915..........	0.97	1.05	9.22	9.94	
January 1916.....	1.27	1.36	13.74	14.70	No. 23, 87, 24, 174
February "	0.54	0.58	12.20	13.09	No. 23, 170A
March "	0.46	0.49	9.77	10.37	No. 170A, 192
April "	0.56	0.59	10.93	11.57	No. 170A, 192
May "	0.56	0.59	10.51	11.11	No. 170A, 191, 192
June "	0.51	0.55	7.72	8.18	No. 87A, 170A, 191, 192
July "	0.66	0.70	8.83	9.21	No. 170A, 191, 192, 270
August "	0.62	0.67	7.54	8.12	No. 87A, 170A, 192
September "	0.73	0.78	7.83	8.37	No. 87A, 23
October "	0.68	0.73	7.31	7.81	No. 87A, 171
November "	0.72	0.79	8.43	9.23	No. 87A, 171
December "	0.59	0.64	9.03	9.78	No. 87A, 171
Year 1916..........	0.65	0.70	9.53	10.19	
January 1917.....	0.67	0.73	9.40	10.36	No. 87A, 171
February "	0.97	1.06	8.19	8.93	No. 87A, 171, 176, 292
March "	0.81	0.89	11.04	12.15	No. 170A, 171, 87A
3 months Year 1917..........	0.80	0.88	9.66	10.61	

* See Table XIV for explanation of numbers.

TABLE XIV

Flotation Oils of Timber Butte Milling Co.

	Producer	Kind of oil
T.B. No. 6	Georgia Pine Turpentine Co	Crude Turpentine L.O.3
T.B. No. 10	United Naval Stores Co.	Pine Oil, M.S. No. 18
T.B. No. 12	Minerals Separation	No. 20 Pine Oil
T.B. No. 21	Georgia Pine Turpentine Co	Carolina Oil of Tar
T.B. No. 23	Georgia Pine Turpentine Co.	Fayetteville Wood Creosote
T.B. No. 24	Georgia Pine Turpentine Co.	Carolina Oil of Tar, Special
T.B. No. 50	United Naval Stores Co.	Pine Oil
T.B. No. 52	United Naval Stores Co.	Turpentine "C"
T.B. No. 80	Pensacola Tar and Turpentine Co.	No. 200 Wood Creosote or C.P. Pine Tar
T.B. No. 85	Pensacola Tar and Turpentine Co.	No. 1000 — Crude Wood Oil
T.B. No. 87	Pensacola Tar and Turpentine Co.	No. 75 Crude Turpentine
T.B. No. 87A	Pensacola Tar and Turpentine Co.	No. 80 Pine Oil, Crude
T.B. No. 88	Pensacola Tar and Turpentine Co.	Pine Oil D.O. Pure
T.B. No. 89	Pensacola Tar and Turpentine Co.	No. 19 Pine Oil
T.B. No. 90	Mackie Pine Products Co.	No. 3 Pine Oil
T.B. No. 110	Standard Oil Company	Cal. Richmond Fuel Oil
T.B. No. 131	Cleveland Cliffs Co.	Refined Hardwood Creosote "XX."
T.B. No. 132	Cleveland Cliffs Co.	Refined Hardwood Creosote No. 2
T.B. No. 140	C. T. Perry and Co.	Oleic Acid
T.B. No. 156	Barrett Manufacturing Co.	No. 4 Coal Tar Creosote.
T.B. No. 157	Barrett Manufacturing Co.	Coal Tar Creosote
T.B. No. 160	Newport Turpentine and Rosin Co.	Pure S.D. Pine Oil
T.B. No. 170	General Naval Stores Co.	No. 9 Pine Tar Oil
T.B. No. 170A	General Naval Stores Co.	No. 1214 Pine Oil
T.B. No. 171	General Naval Stores Co.	No. 5 Pure S. D. Pine Oil
T.B. No. 174	General Naval Stores Co.	No. 8 Pine Tar Oil
T.B. No. 174A	General Naval Stores Co.	No. 814 Pine Tar Oil
T.B. No. 176	General Naval Stores Co.	No. 17 Oil, Hardwood Creosote
T.B. No. 191	Chesapeake Oil Co.	No. 2 Pine Oil
T.B. No. 192	Chesapeake Oil Co.	No. 3 Flotation Oil — Pine Oil
T.B. No. 270	Union Oil Company	Kerosene Acid Sludge
T.B. No. 292	C. G. Betts Company	Refined Coal Tar

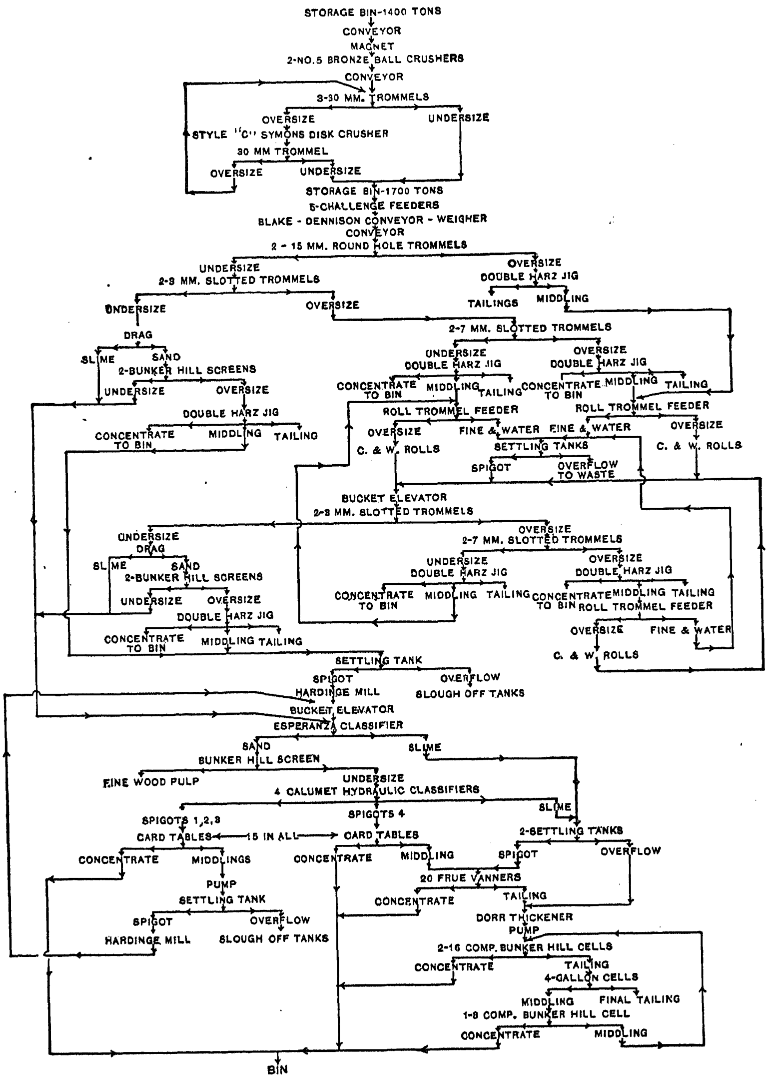


FIG. 55
Flow-sheet, Bunker Hill and Sullivan Mining Co., West Mill No. 2

The West Mill No. 2 of the Bunker Hill and Sullivan Mining Company is described by T. A. Rickard in Mining and Scientific Press, April 10, 1920. The flow-sheet is given in Fig. 55. The ore carries about 10 per cent. lead in the form of galena,

some 2 per cent. zinc in the form of sphalerite, and three to four ounces of silver, in a gangue consisting principally of siderite and quartz. The feed to the flotation machines is at the rate of 95 tons per 24 hours in a pulp containing, after dilution by the return middling, 25 per cent. solids. Pine oil at the rate of 0.08 lb. per ton is used. The concentrate taken off the first flotation machine assays 60 to 65 per cent. lead and the tailing of this machine assays $3\frac{1}{2}$ to 4 per cent. lead. The concentrate from the Callow cells assays 30 per cent. lead and the tailing 1 per cent. lead. The concentrate from the final eight-cell cleaner assays 45 per cent. lead.

Summary. It will be noted that the flow-sheets of the copper mills are, in general, of the primary type. A majority of them employ the rougher-cleaner routing. The flow-sheets of the lead and zinc plants, on the other hand, are generally of the secondary type. A majority of them employ the concentrate-middling or combination routing. Departures from the general routing tendencies are, in many cases, due to causes not apparent on the surface. The Anaconda Copper Co., Copper Plant, is of the concentrate-middling type for the reason that Anaconda smelting operations are built upon the expectation of a concentrate containing a fairly high proportion of iron plus silica, and the relatively low-grade copper concentrate produced by the concentrate-middling routing is, therefore, satisfactory. This same concentrate at the International Smelter at Miami would be wholly uneconomical, and a change to the rougher-cleaner method of routing would undoubtedly be made if the concentrates were to be disposed of to this smelter or another with similar copper requirements. Flotation is a relatively new process and has been tacked on to existing processes in many mills, in which cases structural limitations have sometimes resulted in flow-sheets which depart from the general tendency. Finally, in some cases, caprice and prejudice have played a large part in the installation of flow-sheets which differ widely and unwisely from the general principles set down.

Results to be expected in concentration by flotation depend largely upon the ore. The recovery to be expected increases

with increase in the grade of heads and decrease in the grade of concentrate required. With a copper ore containing from 1 to 2 per cent. copper in the form of sulphide, and no carbonates, a recovery of 85 to 90 or even 95 per cent. may be expected with a concentrate containing not over 20 per cent. insoluble. On a 5 per cent. copper ore of the same variety the recovery should run over 95 per cent. When oxidized copper minerals are present they may be expected to distribute themselves between concentrate and tailing in about the proportions of the ratio of concentration. With lead ores or with zinc ores similar results may be expected, although it is usually not economical to push recovery to the same point as is sought in copper plants. When the feed contains mixed sulphides, the grade of concentrate will be lowered. Where differential flotation is practiced, as is commonly the case with lead-zinc ores and less frequently the case with ores containing copper and iron sulphides, each concentrate will be considerably contaminated with the sulphides sought to be placed in the other concentrate. With lead-zinc ores the lead concentrate ordinarily contains from 40 to 50 per cent. lead and from 8 to say 15 per cent. of zinc and the zinc concentrate from 35 to 50 per cent. zinc and from say 6 to 15 per cent. of lead. Somewhat better results can be obtained by the Horwood process, but whether at a cost which justifies the better grade of products is a matter for individual decision. Separation of copper sulphide from iron sulphide has not been practiced sufficiently to allow any general figures to be set up.

Pulp Formulas

Definitions. "Pulp," in flotation terminology, is a freely-flowing mixture of powdered ore and water. "Pulp density," when the phrase is applied properly, indicates the specific gravity of the pulp. "Percentage of solids" means the ratio, expressed as a percentage, of the weight of the solids in a pulp to the total weight of pulp. "Pulp consistency" is the ratio, by weight or volume as the case may be, between the number of parts of water and the number of parts of solid in a unit of pulp. The number of parts of solid is usually taken as one in the numerical

expression, thus six to one. In the following formulas, the various quantities listed are represented by the letters set after them below: Pulp density, (p); specific gravity of dry solid, (d); specific gravity of water, (1); percentage of solids, (S); pulp consistency by weight, (W), by volume, (w); grams of solid in 100 cc. of pulp, (F); tons of solid per 100 cu. ft. of pulp (T).

Development of formulas.

p = weight in grams of one cc. of pulp.
= weight of water + weight of solid in one cc. of pulp.

Let v = cc. of solid in one cc. of pulp.

Then $1 - v$ = cc. of water in one cc. of pulp.

$$p = 1 - v + vd = 1 + v(d - 1). \tag{1}$$

By definition:

$$W = \frac{1 - v}{vd}. \tag{2}$$

Hence:

$$v = \frac{1}{Wd + 1}. \tag{3}$$

Substituting value of v from (3) in (1):

$$p = 1 + \frac{d - 1}{Wd + 1} = \frac{W + 1}{W + \frac{1}{d}} \tag{4}$$

Solving (4) for d:

$$d = \frac{p}{1 - W(p - 1)}. \tag{5}$$

By definition:

$$W = \frac{100 - S}{S}. \tag{6}$$

Substituting value for W from (6) in (4):

$$p = \frac{100}{100 - \frac{S(d - 1)}{d}}. \tag{7}$$

Solving (7) for d:

$$d = \frac{Sp}{Sp - 100(p - 1)}. \tag{8}$$

Solving (6) for S:

$$S = \frac{100}{W + 1} \tag{9}$$

Solving (8) for S: $$S = \frac{100\, d\, (p - 1)}{p\, (d - 1)}. \tag{10}$$

Substituting in (6) the value of S from (10):

$$W = \frac{d - p}{d\, (p - 1)}. \tag{11}$$

By definition: $$F = \frac{S\, (100\, p)}{100} = Sp. \tag{12}$$

Substituting (12) the value of S from (9):

$$F = \frac{100\, p}{W + 1}. \tag{13}$$

Substituting in (12) the value of S from (10):

$$F = \frac{100\, d\, (p - 1)}{d - 1}. \tag{14}$$

From (1): $$v = \frac{p - 1}{d - 1}. \tag{15}$$

By definition: $$w = \frac{1 - v}{v} = \frac{d - p}{p - 1}. \tag{16}$$

$$\text{Weight, in tons, of 100 cu. ft. of Pulp} = \frac{100\, (62.5)\, p}{2000}.$$

By definition: $$T = \frac{S}{100} \left(\frac{100\, (62.5)\, p}{2000} \right). \tag{17}$$

Substituting value of S from (10) in (17):

$$T = \frac{3.125\, d\, (p - 1)}{d - 1}. \tag{18}$$

Collected Formulas

Pulp density:

$$p = \frac{W + 1}{W + \frac{1}{d}} = \frac{100}{100 - \left(\frac{S\, (d - 1)}{d} \right)}.$$

Sp. Gr. of dry solids:

$$d = \frac{p}{1 - W\, (p - 1)} = \frac{Sp}{Sp - 100\, (p - 1)}.$$

Percentage of solids:

$$S = \frac{100}{W + 1} = \frac{100\, d\, (p - 1)}{p\, (d - 1)}.$$

Pulp consistency by weight:

$$W = \frac{1 - v}{vd} = \frac{100 - S}{S} = \frac{d - p}{d(p - 1)}.$$

Pulp consistency by volume:

$$w = \frac{1 - v}{v} = \frac{d - p}{p - 1}.$$

Grams of solid in 100 cc. of pulp:

$$F = Sp = \frac{100\,p}{W + 1} = \frac{100\,d\,(p - 1)}{d - 1}.$$

Cc. of solid in one cc. of pulp:

$$v = \frac{1}{Wd + 1} = \frac{p - 1}{d - 1}.$$

Tons solid in 100 cu. ft. of pulp:

$$T = \frac{3.125\,d\,(p - 1)}{d - 1}.$$

Figure 18, page 80, presents graphically the relation between S, p and d and is useful for rapid pulp calculations in mill testing.

Figure 56 is useful, especially in connection with microscopic work, for lessening the labor necessary in translating percentage by volume into percentage by weight and *vice versa*.

Metallurgical Formulas

Definitions. The feed or material treated in a metallurgical operation or machine is usually called "heads" or "heading"; the valuable product is called "concentrate"; the impoverished reject is called "tailing"; and any intermediate, unfinished product, insufficiently enriched or insufficiently impoverished, is called "middling." The ratio of the weight of the sought mineral or metal obtained in the concentrate to that present in the heads, expressed as a percentage, is called the "recovery." The ratio of the weight of heads to the weight of concentrate is called the "ratio of concentration." In the following formulas weights and assays of heads, concentrate, tailing and middling are represented by their capital and lower case initial letters

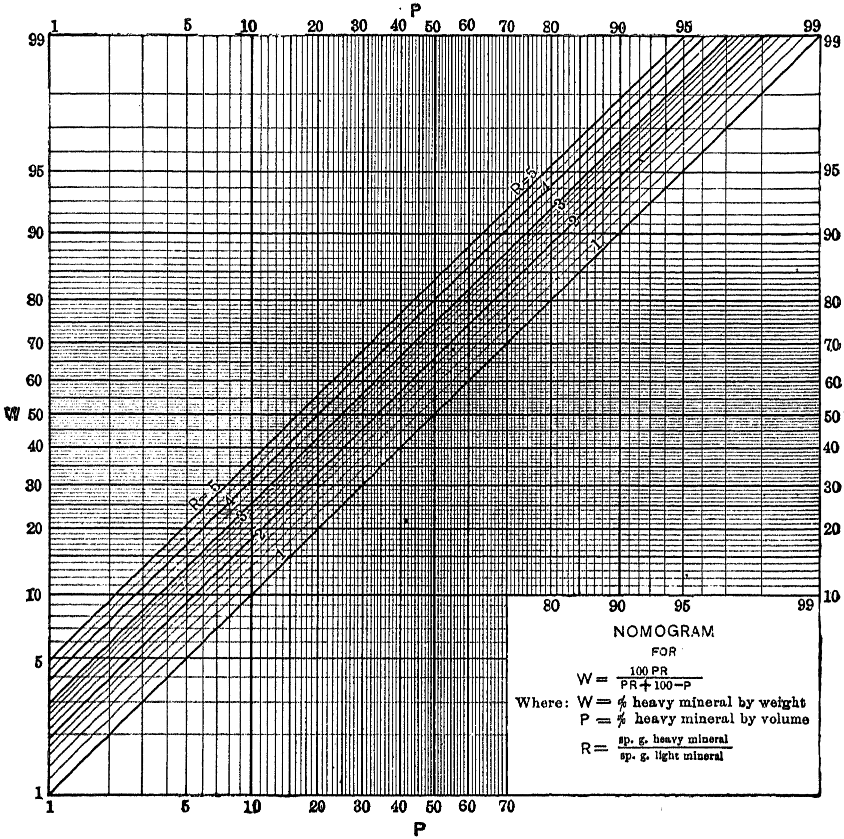


FIG. 56

respectively; recovery is indicated by R and ratio of concentration by K.

Recovery may be determined in any mill operation where concentrate and tailing are the only final products, if the assays only of heads, concentrate and tailing are known. This fact becomes apparent from the following relations:

$$C + T = H. \tag{1}$$

$$Cc + Tt = Hh. \tag{2}$$

By definitions:

$$R = \frac{Cc}{Hh}. \tag{3}$$

Multiply (1) by t: $$Ct + Tt = Ht. \tag{4}$$
Subtract (4) from (2):
$$C\,(c - t) = H\,(h - t). \tag{5}$$
or
$$\frac{C}{H} = \frac{h - t}{c - t}. \tag{6}$$
Substitute the value of $\frac{C}{H}$ from (6) in (3):
$$R = \frac{c\,(h - t)}{h\,(c - t)}. \tag{7}$$

Ratio of concentration may also be expressed in terms of the assays of heads, concentrate and tailing.

Thus, by definition:
$$K = \frac{H}{C}. \tag{1}$$
From (6) in preceding paragraph:
$$\frac{C}{H} = \frac{h - t}{c - t}. \tag{2}$$
Then:
$$K = \frac{c - t}{h - t}. \tag{3}$$

Adjustment of middling. In a laboratory testing operation a middling is usually produced. Clennell * develops formulas for the mathematical disposition of this product as follows:

Assume that, as in practice, the middling will be returned to the same machine or operation; and that the assay of the tailing and concentrate will not be affected by this procedure; and that the middling is not of different mineralogical composition from the original heads. Then, if C, M and T represent the percentage weights of concentrate, middling and tailing respectively; X and Y represent respectively the percentage weights of concentrate and tailing that would be obtained with the middling eliminated, i.e. retreated and thus distributed; and x and y the percentages respectively of the total values in the two final products; the following mathematical relations hold:

* Eng. and Min. Jour., Oct. 30, 1915 and Jan. 1, 1916.

$$C + M + T = X + Y. \tag{1}$$

$$100 = X + Y. \tag{2}$$

$$Cc + Mm + Tt = Xc + Yt. \tag{3}$$

Multiplying (2) by t: $\quad 100\,t = Xt + Yt. \quad (4)$

Subtracting (4) from (3):

$$Cc + Mm + t\,(T - 100) = X\,(c - t) \tag{5}$$

or $$X = \frac{Cc + Mm - t\,(100 - T)}{c - t}. \tag{6}$$

Similarly, multiplying (2) by c, subtracting the resulting equation from (3) and solving for Y:

$$Y = \frac{c\,(100 - C) - Mm + Tt}{c - t}. \tag{7}$$

Now $$x : 100 = Xc : 100\,h \tag{8}$$

Solving (8) for x $$x = \frac{Xc}{h}. \tag{9}$$

Similarly: $$y : 100 = Yt : 100\,h. \tag{10}$$

Solving (10) for y. $$y = \frac{Yt}{h}. \tag{11}$$

Further: $$Xc + Yt = 100\,h. \tag{12}$$

Multiplying (2) by t: $\quad Xt + Yt = 100\,t. \quad (4)$

Subtracting (4) from (12):

$$X\,(c - t) = 100\,(h - t). \tag{13}$$

or $$X = \frac{100\,(h - t)}{c - t}. \tag{14}$$

Similarly: $$Y = \frac{100\,(c - h)}{c - t}. \tag{15}$$

Substituting the value of X from (14) in (9):

$$x = \frac{100\,c\,(h - t)}{h\,(c - t)} \tag{16}$$

By definition of the ratio of concentration, K:

$$K = \frac{X + Y}{X} \tag{17}$$

Substituting values of X and Y from (14) and (15) in (17) and solving for K:

$$K = \frac{c - t}{h - t}. \tag{18}$$

It will be observed that the value of "x" obtained in (14) is the same as the value for R previously obtained and that the value for K just derived is the same as that which was derived for mill operations. In other words, as was to have been anticipated, if the assumptions made at the beginning of the paragraph hold true, equations (6) and (7) give no more information than does equation (18) which is the same as equation (3) of the preceding paragraph, and the "x" in equation (16) is the recovery, R, previously developed.

The assumption made by Clennell that middling obtained in a laboratory testing operation would, in a mill operation under similar conditions, if returned to the head of the machine, be distributed in such a way as not to alter the assays of concentrate and tailing, is not strictly correct in any case. Under certain conditions, however, in flotation testing, it is legitimate to disregard the middling in the calculation of recovery and ratio of concentration. Whether or not the product may be disregarded is to be determined in any given case by a microscopic examination and an assay. If microscopic examination shows that the valuable mineral in the middling is not too coarse to float, nor too large a proportion in the form of included mineral, i.e., not yet freed from gangue, and is not a different mineral, as for instance all chalcopyrite while the original feed contained both chalcocite and chalcopyrite, or an oxidized copper mineral when the feed contained both sulphide and oxide copper; then, if the assay of the middling is not more than twice that of the original feed, or if the bulk of middling is small in relation to the original feed, although the assay is more than twice that of the original feed, the middling may be disregarded in calculating recovery and ratio of concentration. Otherwise the middling should be retreated to determine its behavior. Under any of the contingencies noted it will probably be found that the recovery on middling is poor. If the mineral is coarse, classification in a mechanical classifier followed by tabling of the sands is the solution. If the valuable mineral exists as included grains, regrinding is the only solution and will probably not pay. If the middling is returned it will build up until it must be dis-

charged from circuit either as concentrate or tailing, in which case it will lower the grade of the concentrate or raise the metal content of the tailing. If the middling is of different sulphide content from the heads, i.e., predominantly chalcopyrite, for instance, as compared to a mixture of chalcocite and chalcopyrite in heads, retreatment in a different machine with different flotation agents is the procedure indicated. If the middling is oxidized mineral containing the same metal as the sulphide in the heads, the problem becomes one of the treatment of an oxidized ore, in a different machine, of course.

TABLE XV

Weights and Measures

Weight, English, avoirdupois

Ton short)	Pound (lb.)	Ounce (oz.)	Grain *
1	2 000	32 000	14 000 000
0.000 5	1	16	7 000
0.000 031 25	0.062 5	1	437.5
0.000 000 071 43	0.000 142 86	0.002 285 7	1

The hundredweight (100 lb.), when the short ton (2000 lb.), is used and (112 lb.), corresponding to the long ton (2240 lb.), omitted.

Weight, English, Troy

Ton (short)	Pound (lb.)	Ounce (oz.)	Pennyweight (dwt.)	Grain *
1	2 430.55	29 166.66	583 333.33	14 000 000
0.000 411 43	1	12	240	5 760
0.000 034 285	0.083 333	1	20	480
0.000 001 714 3	0.004 166 7	0.05	1	24
0.000 000 071 43	0.000 173 61	0.002 083 3	0.041 667	1

Weight, apothecaries'

Pound (Troy)	Ounce (℥)	Dram (ʒ)	Scruple ℈	Grain *
1	12	96	288	5 760
0.083 33	1	8	24	480
0.010 42	0.125	1	3	60
0.003 47	0.041 67	0.333 3	1	20
0.000 174	0.002 08	0.016 7	0.05	1

* The grain is the same in avoirdupois, Troy and apothecaries' weights.

Weight, assay. The assay ton = 29,166.66 milligrams. Hence each mg. of metal recovered from an assay-ton charge is equivalent to one Troy oz. per short ton (2000 lb. av.) of the material sampled.

Weight, metric

Ton	Kilogram	Gram	Milligram
1	1 000	1 000 000	1 000 000 000
0 001	1	1 000	1 000 000
0.000 001	0.001	1	1 000
0.000 000 001	0.000 001	0.001	1

The hectogram (=100 grams); the dekagram (=10 grams); the decigram (=0.1 gram); and the centigram (=0.01 gram) are rarely used.

Linear measure, English

Mile	Rod or perch	Yard	Foot	Inch
1	320	1 760	5 280	63 360
0.003 125	1	5.5	16.5	198
0.000 568 19	0.181 82	1	3	36
0.000 189 4	0.060 607	0.333 33	1	12
0.000 015 783	0.005 050 4	0.027 778	0.083 333	1

Linear measure, metric

Kilometer (Km.)	Meter (M.)	Centimeter (cm.)	Millimeter (mm.)	Micron. (μ)	Millimicron. ($\mu\mu$)
1	1 000	100 000	1 000 000	10^9	10^{12}
0.001	1	100	1 000	10^6	10^9
0.000 01	0.01	1	10	10^4	10^7
0.000 001	0.001	0.1	1	10^3	10^6
10^{-9}	10^{-6}	10^{-4}	10^{-3}	1	10^3
10^{-12}	10^{-9}	10^{-7}	10^{-6}	10^{-3}	1

The Myriameter (=10 Kilometers); hektometer (=100 meters); dekameter (=10 meters); and decimeter (=10 centimeters) are rarely used.

Capacity, dry, English

Bushel (bu.)	Peck (pk.)	Quart (qt.)	Pint (pt.)
1	4	32	64
0.25	1	8	16
0.031 25	0.125	1	2
0.015 625	0.062 5	0.5	1

Capacity, liquid, English

Pipe	Hogshead (hhd.)	Tierce *	Barrel (bbl.)	U. S. gallon † (gal.)	Quart (qt.)	Pint (pt.)	Gill
1	2	3	4	126	504	1 008	4 032
0.5	1	1.5	2	63	252	504	2 016
0.25	0.5	0.75	1	31.5	126	252	1 008
0.333 3	0.666 7	1	1.33	42	168	336	1 344
0.007 935	0.015 87	0.023 810	0.031 746	1	4	8	32
0.001 984	0.003 968	0.005 952	0.007 937	0.25	1	2	8
0.000 496	0.001 984	0.002 976	0.003 968	0.125	0.5	1	4
0.000 248	0.000 496	0.000 744	0.000 992	0.031 25	0.125	0.25	1

* The Standard Oil Co. has adopted the tierce, 42 gal., as its barrel, and this practice has been followed by other oil producers and refiners.

† British Imperial gallon = 1.200 91 U. S. gal.

Capacity, metric. The liter (l.) is the unit and is the equivalent of the volume occupied by the mass of 1 kilogram of pure water at maximum density. The smaller units usually employed are the cubic centimeter (cc.) and the cubic millimeter (cu. mm. or $mm.^3$) which are, for all practical purposes 0.001 and 0.000001 liters respectively.

TABLE XVI

Conversion Table

When unit column is entered as

Units	Kilograms	Pounds	Oz. (avoir.)	Gm.	Grains	Mg.	Short tons	Long tons
	read							
	Pounds	Kilograms	Gm.	Oz. (avoir.)	Mg.	Grains	Long tons	Short tons
1	2.204 6	0.453 6	28.350	0.035 274	64.799	0.015 432 4	0.892 87	1.12
2	4.409 2	0.907 2	56.699	0.070 548	129.598	0.030 864 7	1.785 71	2.24
3	6.613 9	1.360 8	85.049	0.105 822	194.397	0.046 297 1	2.678 57	3.36
4	8.818 5	1.814 4	113.398	0.141 096	259.196	0.061 729 4	3.571 43	4.48
5	11.023 1	2.268 0	141.748	0.176 370	323.995	0.077 161 8	4.464 29	5.60
6	13.227 7	2.721 6	170.097	0.211 644	388.794	0.092 594 1	5.357 14	6.72
7	15.432 4	3.175 1	198.447	0.246 918	453.592	0.108 026 5	6.250 00	7.84
8	17.637 0	3.628 7	226.796	0.282 192	518.391	0.123 458 9	7.142 86	8.96
9	19.841 6	4.082 3	255.146	0.317 466	583.190	0.138 891 2	8.035 71	10.08

When unit column is entered as

Units	Short ton	Metric ton (1000 Kg.)	Feet	Meters	Centimeters	Inches	Inches	Millimeters
	read							
	Metric ton (1000 Kg.)	Short ton	Meters	Feet	Inches	Centimeters	Millimeters	Inches
1	0.907 18	1.102 31	0.304 801	3.280 83	0.393 7	2.540 0	25.400	0.039 37
2	1.814 37	2.204 62	0.609 601	6.561 67	0.787 4	5.080 0	50.800	0.078 74
3	2.721 55	3.306 93	0.914 402	9.842 50	1.181 1	7.620 0	76.200	0.118 11
4	3.628 74	4.409 24	1.219 202	13.123 33	1.574 8	10.160 0	101.600	0.157 48
5	4.535 92	5.511 56	1.524 003	16.404 17	1.968 5	12.700 0	127.000	0.196 85
6	5.443 11	6.613 87	1.828 804	19.685 00	2.362 2	15.240 0	152.400	0.236 22
7	6.350 29	7.716 18	2.133 604	22.965 83	2.755 9	17.780 0	177.800	0.275 59
8	7.257 48	8.818 49	2.438 405	26.246 67	3.149 6	20.320 0	203.200	0.314 96
9	8.164 66	9.920 80	2.743 205	29.527 50	3.543 3	22.860 0	228.600	0.354 33

CONVERSION TABLE. — *Continued*

Units	When unit column is entered as: Cu. ft.	Gallons	Cu. ft.	Liters	Gallons	Liters	Kgm. per min.	Tons per 24 hr.
	read							
	Gallons	Cu. ft.	Liters	Cu. ft.	Liters	Gallons	Tons per 24 hr.	Cu. ft. per min.* × θ
1	7.480 5	0.133 68	28.316	0.035 315	3.785 332	0.264 178	1.587 33	1.388 9
2	14.961 0	0.267 36	56.633	0.070 631	7.570 664	0.528 356	3.174 66	2.777 9
3	22.441 6	0.401 04	84.949	0.105 946	11.355 996	0.792 534	4.761 98	4.166 7
4	29.922 1	0.534 72	113.265	0.141 262	15.141 328	1.056 712	6.349 31	5.555 6
5	37.402 6	0.668 40	141.581	0.176 577	18.926 660	1.320 890	7.936 64	6.944 4
6	44.883 1	0.802 08	169.898	0.211 892	22.711 992	1.585 068	9.523 97	8.333 3
7	52.363 6	0.935 76	198.214	0.247 208	26.497 324	1.849 246	11.111 30	9.722 2
8	59.844 2	1.069 44	226.530	0.282 523	30.282 656	2.113 424	12.698 62	11.111 1
9	67.324 7	1.203 13	254.846	0.317 839	34.067 988	2.377 602	14.285 95	12.500 0

Units	When unit column is entered as: Gr. per gal.	Gm. per liter	H.P.	Kw.	Slope in per cent.	In. per ft.	Slope in per cent.	Degrees	In. per ft.	Degrees
	read									
	Gm. per liter †	Gr. per gal.	Kw.	H.P.	In. per ft.	Slope in per cent.	Degrees	Slope in per cent.	Degrees	In. per ft.
1	0.171 185	58.416 23	0.745 7	1.341 0	0.12	8.333	0° 34′	1.746	4° 46′	0.210 0
2	0.342 370	116.832 46	1.491 4	2.682 0	0.24	16.667	1° 9′	3.492	9° 28′	0.418 8
3	0.513 555	175.248 69	2.237 1	4.023 1	0.36	25.000	1° 43′	5.241	14° 02′	0.628 8
4	0.684 740	233.664 92	2.982 8	5.364 1	0.48	33.333	2° 17′	6.993	18° 26′	0.838 8
5	0.855 925	292.081 16	3.728 5	6.705 1	0.60	41.667	2° 52′	8.749	22° 37′	1.050 0
6	1.027 110	350.497 39	4.474 2	8.046 1	0.72	50.000	3° 26′	10.510	26° 34′	1.261 2
7	1.198 295	408.913 62	5.219 9	9.387 1	0.84	58.333	4° 0′	12.278	30° 15′	1.473 6
8	1.369 480	467.329 85	5.965 6	10.728 1	0.96	66.667	4° 34′	14.054	33° 41′	1.686 0
9	1.540 665	525.746 08	6.711 3	12.069 1	1.08	75.000	5° 9′	15.838	36° 52′	1.900 8

* θ = weight per cu. ft. To find cu. ft. per min., divide number from table by θ.
† Gm. per liter × 1000 = parts per million.

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